

TRANSACTIONS
OF THE
AMERICAN INSTITUTE OF MINING
ENGINEERS.

VOL. LII.

CONTAINING THE PAPERS AND DISCUSSIONS OF THE SAN FRANCISCO
MEETING, SEPTEMBER, 1915.

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PREFACE

This volume, the second of the three volumes containing papers and discussions contributed to the Institute in 1915, comprises papers presented at the San Francisco Meeting, September, 1915, and the discussions thereon. For lack of space the papers and discussions relating to iron and steel which were presented at this meeting have been omitted from the present volume and will be published in Vol. LIII, together with papers and discussions on the same subjects presented at the New York Meeting, February, 1916.

Vol. LIII will thus be devoted to contributions on iron and steel and allied subjects.

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Cal., September, 1915

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EXCURSIONS AND SOCIAL FUNCTIONS

The Institute joined with the American Society of Civil Engineers, the American Society of Mechanical Engineers, the American Institute of Electrical Engineers, and the Society of Naval Architects and Marine Engineers, in arranging for a special train from New York to San Francisco to carry members and guests of the five Societies, as well as foreign members of the International Engineering Congress. This train, consisting of Pullman cars and diners, left New York on the evening of Sept. 9. A stop of four hours was made on Sept. 10 for a visit to Niagara Falls. A stop of 10 hours was made at Colorado Springs where arrangements had been made for a trip by automobile through the famous Crystal Park, followed by trips to the summit of Pikes Peak, or visits to gold mills in Colorado Springs. Those members of the party who did not take these trips were taken by automobile to the Cripple Creek district where a visit was made to the mine and mill of Portland Gold Mining Co.

On Sept. 14 the special train stopped for 14 hours at the Grand Canyon, and then reached the City of San Francisco at 9:30 on the evening of Sept. 15. About 150 engineers and guests journeyed by this train and were unanimous in expressing appreciation of the arrangements that had been made by the Joint Committee having this in charge and by the Local Committee at Colorado Springs. Sincere appreciation was also expressed by the 30 members of the party who made the visit to the Cripple Creek district as guests of the Portland Gold Mining Co.

The Joint Committee on Arrangements of the five Societies was as follows:

ALEX. C. HUMPHREYS, *Chairman*
 CHARLES WARREN HUNT, *Secretary*
 WILLIAM L. SAUNDERS
 WILLIAM H. WILEY
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 AMBROSE SWASEY
 CALVIN W. RICE

Excursions in San Francisco.—The Local Committee representing the National Engineering Societies and the International Engineering Congress had arranged a number of excursions in San Francisco for members of the various Societies involved, of which the following is a list:

Sept. 18: The San Francisco High Pressure Fire System, the Potrero Gas Works, and Electric Station "A," Pacific Gas & Electric Co.

Sept. 18: Spring Valley Water Works Properties on east side of San Francisco Bay.

Sept. 19: Spring Valley Water Works Reservoirs and Pumping Stations on San Francisco Peninsula.

Sept. 19: The Delta Lands of the Sacramento & San Joaquin Rivers.

Sept. 17-18: Great Western Power Co.'s Hydro-electric Development on Feather River and Dredging at Oroville.

Sept. 18-19: Pacific Gas & Electric Co.'s Hydro-electric Development at Lake Spaulding and Drum Power House, and the Gold Mines at Grass Valley.

Sept. 17-19: Oil Fields at Coalinga.



BRONZE MEDAL PRESENTED TO THE INSTITUTE AT THE PANAMA-PACIFIC
INTERNATIONAL EXPOSITION

Boat Trip Around San Francisco Bay.—Through the courtesy of Colonel D. C. Jackling, the Local Committee of the Institute had provided for a boat trip around San Francisco Bay for members of the Institute and their guests only, on the beautiful and commodious yacht of Colonel Jackling, which holds comfortably 200 persons. Only 200 tickets were issued and every one of these was eagerly sought after. A buffet luncheon and afternoon tea were served on board the yacht; the trip included a visit of about an hour's duration to the smelting plant of the Selby Smelting & Lead Co. There does not exist a better opportunity for promoting acquaintance among a large number of people than by means of an attractive boat trip of this character and it is safe to say that every member of the party enjoyed himself or herself to the utmost. This trip started at 10:00 a.m. on Saturday, Sept. 17, and all the guests were on shore again by 4:30 in the afternoon.

Garden Party of Charles Butters.—On Sunday afternoon, Sept. 19, members and guests of the Institute were entertained by Mr. and Mrs. Charles Butters at a garden party in their beautiful place in Berkeley. This event proved to be one of the most enjoyable of the entire meeting.

Banquet.—A complimentary banquet was tendered to visiting members and guests by the San Francisco members on Friday evening, Sept. 17 at 7:30 p.m., at the Palace Hotel. This banquet was attended by about 400 persons. E. H. Benjamin acted as toastmaster and toasts were responded to by the following speakers:

BENJAMIN IDE WHEELER: An Engineer
 W. L. SAUNDERS: California
 VAN H. MANNING: Cooperation
 GEORGE OTIS SMITH: The Outlook
 T. A. RICKARD: The Ladies

Luncheons.—Thursday, Sept. 16, luncheon was had at Bellevue Hotel where the sessions were held. On Friday there was a joint subscription luncheon of the Institute and the American Electrochemical Society at the Clift Hotel, adjoining the Bellevue Hotel.

Entertainment of Ladies.—On Wednesday evening, Sept. 15, the Ladies' Committee held an informal reception for the visiting ladies and members in the lobby of the Bellevue Hotel. This reception began shortly after the arrival of the Engineers Special Train from the East. On Thursday, Sept. 16, the ladies were entertained by an automobile trip through the famous Golden Gate Park and the Presidio, after which a complimentary luncheon was served on the Exposition grounds; the ladies of San Francisco acted as guides to the visiting ladies in seeing the principal objects of interest in the Exposition. On Friday morning the ladies were taken through Chinatown, a favorite trip of all visitors to San Francisco.

AMERICAN INSTITUTE OF MINING ENGINEERS' DAY AT THE EXPOSITION

The management of the Panama-Pacific International Exposition had designated Friday, Sept. 17, as the American Institute of Mining Engineers' Day at the Exposition and in the afternoon at 4 o'clock ceremonies were held by the officers of the Exposition and the officers of the Institute in the Court of Abundance. These ceremonies consisted in the presentation to the Institute of a bronze medal (see accompanying illustrations). The presentation was made on behalf of the Exposition officers by Director F. L. Brown, in a felicitous speech, thanking the Institute for its share in the activities of the Exposition by holding its meeting in San Francisco

and joining in the support of the International Engineering Congress in San Francisco. President Saunders responded on behalf of the Institute in the following words:

It is said that the last words of William IV on hearing the sound of cannon on the anniversary of the Battle of Waterloo, were, "That was a great day for England;" and so as the members of this Institute look out upon this great exposition and see and hear what California, a mining State, has done in but little more than a century, we may perhaps be pardoned for exclaiming with joy and pride, This is a great day for the American Institute of Mining Engineers.

We are glad to be here, we are glad that we are alive: happy in the thrill of vitality which is quickened by California spirit and by the elixir of your glorious climate. On behalf of the Institute which I represent, I thank you for giving us this day. In thus honoring the Institute you are paying a tribute of regard and appreciation to the whole mining industry; to the lusty pioneer who blazed the trail and whose dry bones were the only monuments to mark the milestones of his progress: to the educated engineer and the capitalist, whose skill and money were spent beneath the soil in daring efforts to release the hidden forces of Nature and put them to the service of mankind. You are paying tribute to the geologist, the chemist, and the metallurgist, for they too are mining engineers and members of this Institute.

I speak of California as a mining State; as such were you born. You received your baptism in the waters of the South Fork of the American River, when Marshall discovered gold at Sutter's Mill in 1848. This was the beginning of the golden age for California and the world. The golden age means more than the age of gold; it is the age of progress, of prosperity and industrial renown and wealth; the age of railways and steamships, of manufacture, of the steam engine, of electricity, of the telephone, telegraph, the automobile and the aeroplane. All these things were made possible, yes, they were initiated and installed through means which were afforded by the mining industry. The mining engineer has multiplied and replenished the earth through subduing it; he has released those forces of nature which have been hidden through countless ages; he has through the science of metallurgy and chemistry made it economically possible to widen and enlarge to enormous proportions the use and influence of these forces. Look into the modern locomotive and steamship and we shall see that almost every pound of their vital structure is a product of the mines: The lathe that fashioned the shaft is itself a product of the mine and it gets its power to turn from coal mined by machines which are themselves products of the mines. Lest you say to all this that the art of mining is as old as the hills let us remind you of the fact that mining at low cost, mining low-grade ore, and doing these things on a large scale and the art of reduction and treatment through metallurgy, are the accomplishments of the mining engineer of our time. This was the key which unlocked those forces which have been the impulse behind the great industrial progress of the world. This too has all been accomplished since the discovery of gold in California.

Let us review some of the facts of history. The first half of the nineteenth century was a period marked by little progress in industrial wealth. The world's output of gold from 1800 to 1835 varied from \$10,000,000 to \$15,000,000 annually. The yearly product of iron during this period was only between 2,000,000 and 3,000,000 tons, coal less than 50,000,000 tons, copper less than 50,000,000 *pounds*. There were no transcontinental railways, no great steamships, no battleships, no factory system of manufacture, no large corporations. The total gold product of the United States during the period mentioned scarcely reached \$1,000,000 annually. Beginning about the middle of the century when California set the pace by adding in a single year \$50,000,000, and even \$80,000,000 to the gold of the world, there followed a steady and increasing progress made in the output of iron, coal, and copper. Railway building progressed on almost parallel lines, so too did the deposits in the banks of the United States. It is of interest to note here that the American Institute of Mining Engineers got aboard in 1871 and thereafter shared closely in growth and prosperity this march of industrial progress.

The world's production of gold at the present time is over \$450,000,000 annually. The United States alone now produces over 26,000,000 tons of pig iron annually, and about 550,000,000 tons of coal. The world's yearly production of copper is now over 900,000 tons and of this the United States produced over 500,000 tons. So too we find that the average per capita wealth of the people of the United States has risen from \$300 in 1850 to over \$1,300. The influence of mining on wealth is conspicuously shown in the case of your neighbor, the State of Nevada, where the per capita wealth of her people is about \$4,800. Nevada is strictly speak-

ing a mining State, with easy divorce laws to interest and allure the homeless and distracted mining engineer.

California has added more than \$1,600,000,000 to the gold supply of the world. Her record average of over \$25,000,000 in gold annually for 50 years is unprecedented. It has been estimated that the actual value of gold deposits should be multiplied eight times to get the true value in credit and capital: If this is true then California has in her gold supply alone contributed over \$12,000,000,000 to the capital wealth of the United States.

Gold is the standard of all values. It is the measure of credit, the basis of exchange. "Change and decay in all around I see," but in gold there is no change, no decay. The lure of gold has discovered continents and turned deserts into fertile lands. Gold was the beacon light which led Columbus across the Atlantic. Cortez, Pizarro, Balboa, and others who are called explorers were really pioneers and prospectors whose voyages led to the early development of mining in Peru and Chile. But not till California led the way did the mine explorer become a world builder; he was the advance agent of prosperity.

The stability of this country in peace and in war is due mainly to the mining engineer. Our industrial strength comes from the mines, the mills, and the furnaces which are now so well organized on a peace basis and which have in their substance all the sinews of war. Military supremacy comes from the same elements as industrial supremacy. It is mainly a question of organizing the resources of the country. Of one thing we may feel sure and that is that no nation can ever make a scrap of paper out of a gold eagle.

You may crush, you may shatter the coin if you will,
But the value of gold dust remains with it still.

It was because of the recognized importance of the mining and metallurgical industry that the American Institute of Mining Engineers was organized at Wilkes-Barre, Pa., in 1871. Following the War of the Rebellion and coincident with the development of the mining industry in the far Western States, there arose a great demand for mining engineers. It was recognized that the chief factor in the development of the western country was the mining industry; it created a romantic interest and it gave a stimulus to the people in general and even to Congress which granted large subsidies for railway building. Scientific schools were established and technical journals disseminated engineering literature; but the mining men, those actually engaged in the mines and mills and smelters, were not organized. They needed a common ground for the exchange of ideas, a forum for discussion and education; and so the Institute was established. It was the second society of its kind following the organization of the American Society of Civil Engineers. The *Transactions* of the Institute cover geology, assaying, milling, smelting, mining, quarrying, and general construction management. The papers submitted at the meetings cover a wide field and are recognized by engineers and scientists throughout the world as of significance and value.

The Institute is incorporated under the laws of the State of New York, organized and existing with the object of promoting the arts and sciences connected with the economic production of the useful minerals and metals and the welfare of those employed in these industries by means of meetings for social intercourse and the reading and discussion of professional papers, and to circulate by means of publications among its members the information thus obtained. Its membership includes those who have worked their way up from the ranks as well as graduates from scientific schools and colleges. Leading professors and technical educators have always been prominent in its membership. The welfare and safety of the mine workers is a subject of recent and earnest interest among the members, and a new field of usefulness has been developed in using the power and the influence of the Institute wherever the industry represented by mining and metallurgy or the people engaged in the work may be benefited through State or national legislation.

On behalf of the members of the American Institute of Mining Engineers, numbering now over 5,000 and scattered throughout the world, I extend to President Moore, to San Francisco, and to California, a full measure of warm congratulations and our best wishes for continued and increasing health, happiness, and prosperity.

TECHNICAL SESSIONS

The first technical session was held at 10:00 a.m. on Thursday, Sept. 16, 1915, in the Palm Room of Bellevue Hotel, Charles W. Merrill presiding.

An address of welcome on behalf of California was made by Arthur Arlett to whom Gov. Hiram Johnson deputed this duty. President Saunders replied on behalf of the Institute as follows:

On behalf of the American Institute of Mining Engineers it gives me pleasure to express our deep appreciation of the privilege extended to us of holding this meeting in California. We are glad to be here. It is always an inspiration and a pleasure to inhale your life-giving air. It is always a rare treat to get the true spirit of the West by rubbing elbows with your people.

But this meeting seems to me to have an importance, a far-reaching significance beyond any other that has been held since the foundation of the Institute. I am also impressed by the fact that it has a great significance to the people of California. In the busy march of your industrial life, with the many activities which now flourish engrossing your attention and giving wealth, prosperity and happiness to your people, does it occur to you that after all the basic foundation upon which your whole structure lies is the mining industry. Mining made California what she is and mining is still your greatest industry. Your mines have made it possible to establish the gold standard throughout the world. Your mines first created a human movement toward the far West which not only gave you those young and lusty pioneers of industry, but they peopled the western part of this country. It was the romantic mining spirit of California which influenced so successfully the building of railways to the far West.

California created the mining engineer, and the mining engineer, true to his parent, has made California what she is. He did more than this, your scientific schools and your many workshops in the field sent forth the men who first developed the mining industry in America, Australia, Africa and other places of the world. Others have done much but the men of California blazed the first trail and did that part of the work which is the hardest of all, the work of the pioneer.

This Institute is the only organized body which for 44 years has represented the mining men, from the man with the pick, who rises to a higher charge, to the college man, the geologist, the chemist, the assayer, the engineer, the metallurgist and the manager. Our work is coördinated for the safety, welfare and profit of the industry.

We feel that we have a right to be here at this festival held in honor of your prosperity, and as we have shared with you some of the burdens and hardships of the past, you will I trust permit us at this happy time also to share with you at least some measure of your pride and rejoicing.

The following papers were then presented by their authors or authors' representatives:

- Underground Mining Systems of Ray Consolidated Copper Co. By Lester A. Blackner. (Discussed by Sidney J. Jennings.)
Some Problems in Copper Leaching. By L. D. Ricketts. (Discussed by Frederick Laist, T. T. Read, E. H. Hamilton.)
Notes on Homestake Metallurgy. By Allan J. Clark. (Discussed by G. H. Clevenger, J. A. Fulton.)
The Metallurgy of Gold in the Witwatersrand District, South Africa. By F. L. Bosqui. (Discussed by Sidney J. Jennings, C. W. Merrill, Charles Butters, Hennen Jennings, Allan J. Clark and written discussion by W. A. Caldercott.)

On the afternoon of Thursday, Sept. 16, 1915, at 2:00 p.m., a Session on Gold and Silver was held in the Palm Room of Bellevue Hotel, F. Lynwood Garrison presiding.

The following papers were then presented by their authors or authors' representatives:

- Mill and Cyanide Plant of Chiksan Mines, Korea. By Charles W. De Witt.
Slime Agitation and Solution Replacement Methods, West End Mill, Tonopah, Nev. By Jay A. Carpenter.

- Cyaniding Practice of Churchill Mining Co., Wonder, Nev. By E. E. Carpenter.
 The Tonopah Plant of the Belmont Milling Co. By A. H. Jones.
 A Rule Governing Cupellation Losses. By W. J. Sharwood. (Written discussion by Frederic P. Dewey.)
 Zinc-Dust Precipitation Tests. By Nathaniel Herz.
 Recovery of Mercury from Amalgamation Tailing, Buffalo Mines, Cobalt. By E. B. Thornhill. (Discussed by D. B. Huntley, H. G. S. Anderson.)

The following papers were read by title only:

- Amalgamation Tests. By W. J. Sharwood.
 Electric Furnace for Gold Refining at the Alaska-Treadwell Cyanide Plant. By W. P. Lass. (Written discussion by R. M. Keeney.)

On the afternoon of Thursday, Sept. 16, 1915, at 2:00 p. m., a Session on Geology and Mineralogy was held in the Red Room of Bellevue Hotel, Andrew C. Lawson presiding.

The following papers were then presented by their authors or authors' representatives:

- † The Geology of the Iron-Ore Deposits in and Near Daquiri, Cuba. By James F. Kemp. (Discussed by W. E. Pratt.)
 The Occurrence of Covellite at Butte, Mont. By A. Perry Thompson. (Discussed by A. S. Eakle, H. W. Turner, C. F. Tolman, A. F. Rogers, L. C. Graton, A. C. Lawson, J. C. Ray.)
 † The Formation and Distribution of Bog Iron Ores. By C. L. Dake. (Discussed by A. C. Lawson.)
 Geology of the Burro Mountains Copper District, New Mexico. By R. E. Somers. (Discussed by J. B. Umpleby, L. C. Graton.)
 Method of Making Mineralogical Analysis of Sand. By C. W. Tomlinson.
 The Copper Deposits of San Cristobal, Santo Domingo. By Thomas F. Donnelly. (Written discussion by F. Lynwood Garrison.)

The following papers were read by title only:

- † Additional Data on Origin of Lateritic Iron Ores of Eastern Cuba. By C. K. Leith and W. J. Mead.
 The Formation of Oxidized Ores of Zinc from the Sulphide. By Y. T. Wang.
 † The Formation and Distribution of Residual Iron Ores. By C. L. Dake.

On Friday, Sept. 17, 1915, at 9:00 a. m., a Session on Petroleum and Gas was held in the Red Room of Bellevue Hotel, Arthur F. L. Bell presiding.

The following papers were then presented by their authors or authors' representatives:

- Petroleum as Fuel Under Boilers and in Furnaces for Heating, Melting, and Heat Treatment of Metals. By W. N. Best. (Discussed by A. F. L. Bell, William A. Williams, Hennen Jennings, D. T. Day, Mark L. Requa.)
 Gasoline from "Synthetic" Crude Oil. By Walter O. Snelling. (Discussed by A. C. McLaughlin, William A. Williams, W. N. Best, Mark L. Requa, A. F. L. Bell, D. T. Day, Walter Stalder.)
 The Possible Occurrence of Oil and Gas Fields in Washington. By Charles E. Weaver. (Discussed by A. C. McLaughlin, W. N. Best, Milnor Roberts, W. E. Pratt, D. T. Day, George O. Smith.)
 Protecting California Oil Fields from Damage by Infiltrating Water. By R. P. McLaughlin. (Discussed by A. C. McLaughlin, M. E. Lombardi, A. F. L. Bell, William A. Williams, M. L. Requa, C. D. Keen.)
 The Cost of Maintaining Production in California Oil Fields. By M. E. Lombardi. (Discussed by M. L. Requa, C. D. Keen, H. M. Bacon.)
 Important Topping Plants of California. By Arthur F. L. Bell. (Discussed by William A. Williams, D. T. Day.)

The following papers were presented by title only:

- The Occurrences of Petroleum in the Oil Fields of Eastern Mexico as Contrasted with those of Texas and Louisiana. By E. T. Dumble.

† Held for publication in vol. liii.

- Oil, Gas and Water Content of Dakota Sand in Canada and United States. By L. G. Huntley.
 Correlation and Geological Structure of the Alberta Oil Fields. By D. B. Dowling.
 The Mexican Oil Fields. By L. G. Huntley.
 Sliding Royalties for Oil and Gas Wells. By Roswell H. Johnson. (Discussed by William A. Williams, A. C. McLaughlin, A. F. L. Bell.)
 The Furbero Oil Field, Mexico. By E. DeGolyer.

On Friday, Sept. 17, 1915, at 9:00 a.m., Session on Electro-Metallurgy Joint Session with the American Electrochemical Society, in the Palm Room of Bellevue Hotel, Lawrence W. Addicks, President of the American Electrochemical Society, presiding.

The following papers were presented by their authors or authors' representatives:

- † The Thermal Insulation of High-Temperature Equipment. By Percy A. Boeck. (Discussed by Lawrence Addicks, J. W. Beckman, A. H. Kreiger, J. W. Richards, Kenneth Seaver.)
 Metallurgical Industries as Possible Consumers of Electric Power. By Dorsey A. Lyon and Robert M. Keeney. (Discussed by Lawrence Addicks, G. H. Clevenger, C. H. Booth, Kenneth Seaver, J. W. Richards, J. W. Beckman, T. T. Read.)
 * The Electric Precipitation of Gold, Silver and Copper from Cyanide Solutions. By G. H. Clevenger. (Discussed by Dr. Daschak, J. W. Richards, T. T. Read.)
 * Solution Stratification as an Aid to the Purification of Electrolytes. By Francis R. Pyne. (Discussed by Frederick Laist, Lawrence Addicks, J. W. Richards.)
 Roasting and Leaching Concentrator Slimes Tailings. By Lawrence Addicks. (Discussed by Frederick Laist, Lawrence Addicks, J. W. Richards.)
 * The Electrolysis of Copper Sulphate Liquors, Using Carbon Anodes. By Lawrence Addicks. (Discussed by L. D. Ricketts, E. H. Hamilton, Frederick Laist, Charles Butters.)
 Hydro-Electrolytic Treatment of Copper Ores. By Robert Rhea Goodrich.

The following papers were read by title only:

- * Melting of Ferro-Alloys in the Electric Furnace. By R. S. Wile.
 * Radiography of Metals. By Wheeler P. Davey.

On Friday afternoon, Sept. 17, 1915, at 1 o'clock, a Session on Mining, Milling and Non-Ferrous Metallurgy, was held in the Palm Room of Bellevue Hotel, Karl Eilers presiding.

The following papers were presented by their authors or authors' representatives:

- Tramming and Hoisting at the Copper Queen Mine. By Gerald Sherman.
 Ventilation of the Copper Queen Mine. By Charles A. Mitke. (Discussed by Gerald Sherman, J. P. Hodgson.)
 Fire-Fighting Methods at Mountain View Mine, Butte, Mont. By C. L. Berrien.
 Standardizing Rock-Crushing Tests. By M. K. Rodgers.
 The Concentrator of the Timber Milling Co., Butte, Mont. By Theodore Simons.
 The British Columbia Copper Co.'s Smelter, Greenwood, B. C. By Frederic K. Brunton.
 The Advantages of High-Lime Slags in the Smelting of Lead Ores. By S. E. Bretherton. (Written discussion by A. Eilers.)

The following papers were read by title only:

- The Application of the Apex Law at Wardner, Idaho. By Fred T. Greene.
 Mine Pumping. By Charles Legrand.
 Mining Conditions on the Witwatersrand. By W. L. Honnold.
 Churn-Drilling Costs, Sacramento Hill. By Arthur Notman.

* Not included in this volume; to be published in the *Transactions* of the American Electrochemical Society.

† Held for publication in vol. liii.

- †† The Stresses in the Mine Roof. By R. Dawson Hall.
 Kick vs. Rittinger. By Arthur O. Gates. (Written discussion by Algernon Del Mar.)
 † Rotary Desulphurizing Kilns. By S. E. Doak.
 Hardinge Mill Data. By Arthur F. Taggart.
 The Salida Smelter. By F. D. Weeks.
 Lead Smelting at El Paso. By H. F. Easter.
 The Mellen Rod-Casting Machine. By R. C. Patterson, Jr.
 Conveyor-Belt Calculating Chart. By J. D. Mooney and D. L. Darnell.

On Friday afternoon, Sept. 17, 1915, at 1 o'clock, a Session on Iron and Steel was held in the Red Room of Bellevue Hotel, Joseph W. Richards presiding.

The following papers were presented by their authors or authors' representatives:

- † Manufacture and Tests of Silica Coke-oven Brick. By Kenneth Seaver. (Discussed by William A. Williams, J. W. Richards.)
 † Iron Ores of California and Possibilities of Smelting. By C. Colcock Jones. (Discussed by D. A. Lyon, J. W. Booth, Bradley Stoughton, H. W. Lash, J. W. Richards.)
 † The Electric Furnace in the Foundry. By William G. Kranz. (Written discussion by M. Petinot.) (Discussed by J. W. Booth, J. W. Richards, H. W. Lash.)
 † Commercial Production of Sound Homogeneous Steel Ingots and Blooms. By E. Gathmann. (Discussed by H. W. Lash, J. W. Richards.)

The following papers were read by title only:

- † Conversion Scale for Centigrade and Fahrenheit Temperatures. By Hugh P. Tieman. (Discussed by J. W. Richards.)
 † The Duplex Process of Steel Manufacture at the Maryland Steel Works. By F. F. Lines.
 † Suggestions Regarding the Determination of the Properties of Steel. By A. N. Mitinsky. (Written discussion by H. M. Howe, Mansfield Merriman, Gaetano Lanza, Lawford H. Fry, H. V. Wille, James E. Howard, George K. Burgess.)

* Held for publication in vol. liv.

† Held for publication in vol. liii.

PAPERS

Notes on Homestake Metallurgy

BY ALLAN J. CLARK,* E. M., LEAD, S. D.

(San Francisco Meeting, September, 1915)

It is nearly three years since the metallurgy of the Homestake ore was discussed with considerable thoroughness, in a paper¹ read before the Institution of Mining and Metallurgy.

Certain changes have been made in this period which are perhaps of sufficient interest to justify a brief description, and it is chiefly with such details and miscellanies that this paper will deal. In the circumstances, a certain amount of repetition is inevitable, but matter treated of in the former paper will be referred to here only when it is necessary for the sake of coherency. For full descriptions of equipment and technique, the former paper should be consulted.

The mineralized slates and schists which constitute the ore vary considerably in composition, but the unoxidized ore, perhaps constituting the major part of the reserves, contains either chlorite or hornblende (cummingtonite), with quartz, carbonates of lime, magnesia and iron, and arsenopyrite, pyrite, and pyrrhotite. Ferrous minerals predominate, and this fact has been an important factor in determining the metallurgical treatment.

With one or two exceptions the minerals noted are of relatively high specific gravity, and the ore as a whole is exceptionally heavy, many determinations giving an average specific gravity of 3.00.

This high specific gravity presents one decided advantage, when the cost of treatment is compared with operations elsewhere, in that the ton, almost universally the basic unit, represents a volume probably less by 10 per cent. than that of many gold ores. On the other hand, this high gravity renders more difficult the discharge of pulp from mortars, its distribution on amalgam tables, and its transportation in launders.

In the same way, as attesting the usual balance of such variations of different ores, it may be noted that the needle-like fibers of hornblende (cummingtonite), interlacing throughout the mass, render the ore more difficult to crush than would be anticipated, but on the other hand assist

* Metallurgist, Homestake Mining Co.

¹ Clark and Sharwood: The Metallurgy of the Homestake Ore, *Transactions of the Institution of Mining and Metallurgy*, vol. xxii, p. 68 (1912-13).

in maintaining free leaching in both sand and slime treatment, retaining their characteristic form as far as they can be traced with the microscope.

The metallurgical equipment consists of:

At the South Side:

- 3 stamp mills (660 stamps) with 36,356 sq. ft. of amalgam plates.
- 1 regrinding plant, with independent cone system and 540 sq. ft. of amalgam plates.
- 4 batteries of cones for classification.
- 3 clarifying-tank houses.
- 1 sand plant.

At the North Side:

- 2 stamp mills (360 stamps).
- 2 tank houses.
- 2 cone houses.
- 1 sand plant.

At Deadwood:

- 1 slime plant, treating the combined slime from South and North sides.

A flow sheet is given in Fig. 1.

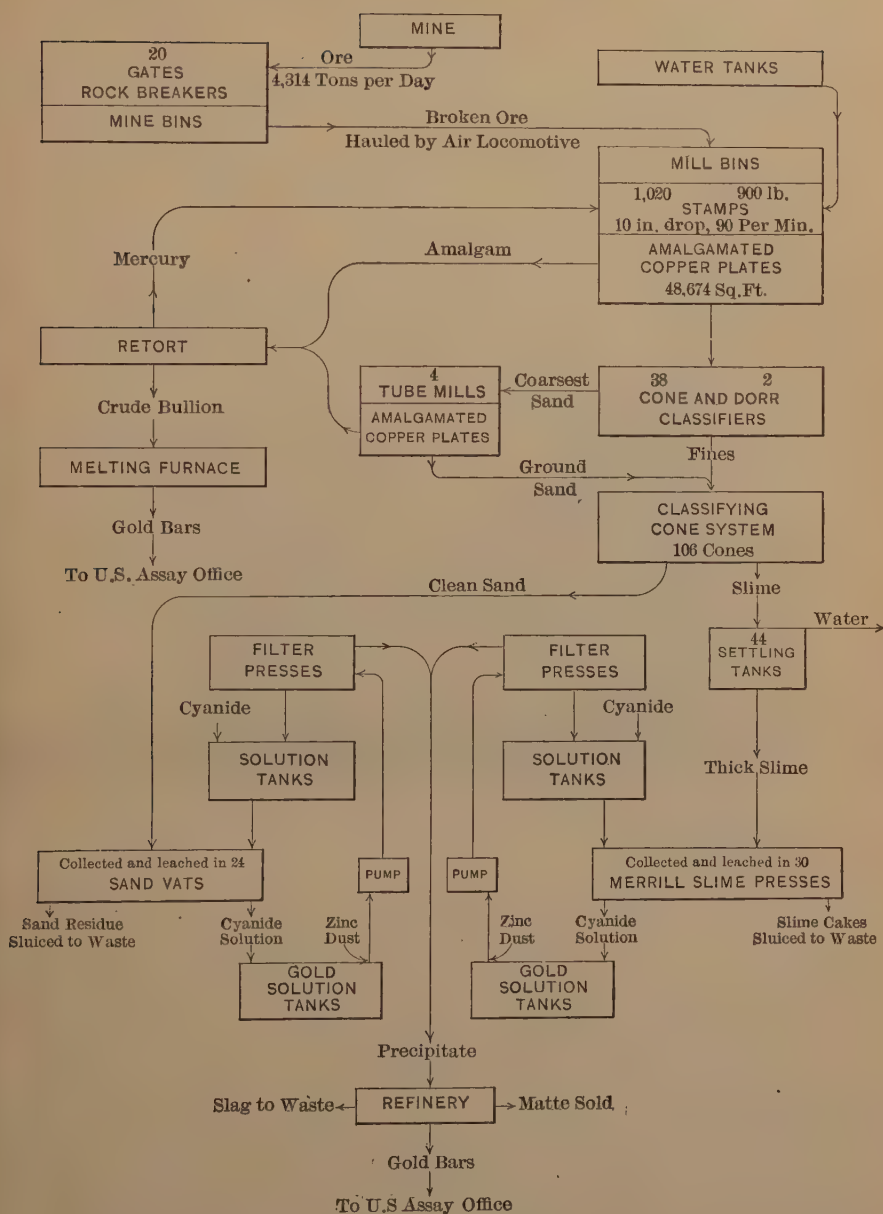
The ore supplied to the North Side mills is usually drawn from the upper levels of the mine, and is at least partly oxidized. It is more easily penetrated by cyanide solutions than is the unoxidized ore, and satisfactory extractions are at present made from the sandy portion of the mill tailing without further reduction in tube mills. In almost all other respects, practice at the two divisions of the works is identical, and unless otherwise stated further description will be of operations at the southern branch.

STAMP MILLING

The stamps, when newly shod, weigh 900 lb. and drop 10 in., making 88 drops per minute. Pulp discharges through No. 8 diagonal needle-slot screens. Inside amalgamation is practiced, quicksilver being fed to the mortar, and a copper chuck-block, about 5½ in. wide, being placed inside the mortar, below the discharge screen. To facilitate discharge and distribution of the pulp over the amalgam plates, water is liberally used, the usual ratio being from 10 to 11 parts, by weight, to one of solids.

Electric drive, now in use for more than two years, permits more nearly continuous operation than formerly. A 25-hp. back-geared motor drives each 10 stamps, transmitting motion by a 16-in. belt.

Chuck-blocks are cleaned in rotation, the usual interval between cleanings being about two weeks; outside plates, except those of the fourth row, are cleaned and dressed daily. Fourth-row plates are dressed at intervals of two days. At the foot of the rows of plates are traps, from which the sands are removed every second or third day. These trap sands are run over a special silvered amalgam plate, the tailing rejoining the main stream of tailing. It was formerly the practice to clean these traps



daily, but this is not necessary since the development of the regrinding plant, which acts, in effect, as a large trap for all the mills.

Amalgam is retorted three times each month. Oil-fired retorts are used, the resulting bullion being melted in coke-fired furnaces. The loss of quicksilver in retorting is almost *nil*; the subsequent melting loss, which averages slightly more than 1 per cent. of the weight of the crude bullion, includes some quicksilver not driven off in the retorts.

A new mortar (Fig. 2) has recently replaced the familiar design. In this newer design the old "inside lines," remarkably effective in main-

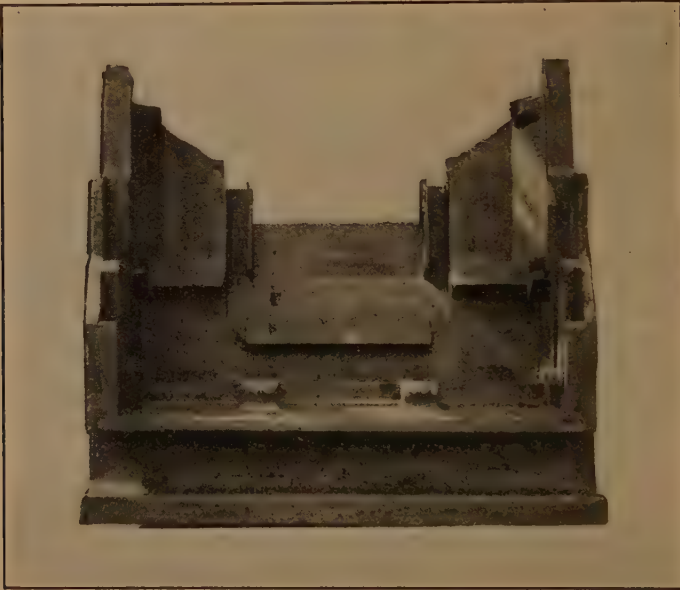


FIG. 2.—NEW OPEN-FRONT MORTAR IN USE AT HOMESTAKE MILLS.

taining a rapid discharge, have been retained. All the front above the screen rest has been cut away, as has also a considerable portion of the back. This has given greater accessibility, with resulting safety to the men when engaged in changing iron and other necessary operations; it permits the use of longer bossheads and simplifies the care of feeders, since an observer at the front of the mortar can determine whether the ore is feeding properly. The slight tendency to splash at the back is corrected by a false back, or apron, of canvas; a small chip tray of 4-mesh screen, resting above the discharge screen, impounds floating particles of wood, which are removed from time to time, washed, and burned. The ashes, which assay as high as \$300 per ton, are set aside to accumulate in sufficient quantity and are eventually sold.

The unoxidized ore, although carrying large quantities of sulphides,

amalgamates well, but not so freely as old records show to have been the case in earlier years, when less of this class of ore came to the mills. A longer time is apparently required to properly incorporate the quicksilver with the pulp, and the resulting amalgam is recovered somewhat farther from the battery. The results given in Table I illustrate this tendency.

TABLE I.—*Comparison of Amalgamation Results on Ores of Different Percentages of Sulphides*

Column I. All mills, three-month period in 1910.

Column II. One mill, crushing unoxidized ore exclusively, three months, 1915.

Of the Total Recovery by Amalgamation, there was Recovered from:	I. Per Cent.	Cumulative Per Cent.	II. Per Cent.	Cumulative Per Cent.
Batteries.....	49.4	49.4	35.6	35.6
First-row plates.....	37.5	86.9	42.9	78.5
Second-row plates.....	4.4	91.3	8.1	86.6
Third-row plates.....	2.3	93.6	3.5	90.1
Fourth-row plates ^a	0.9	94.5	2.0	92.1
Skimmings.....	4.0	98.5	6.9	99.0
Trap sands.....	1.5	100.0	1.0	100.0
Percentage of ore value.....		71.7		69.7

^a Only two mills (440 stamps) equipped with a fourth row of plates.

The "skimmings" of the above table are recovered from re-treatment of the foul amalgam, sulphide particles, etc., removed from the cleanup sink during the cleanup of the chuck-block and plate amalgam.

These cleanings are re-treated in a small barrel 24 in. in diameter by 30 in. long, in which iron balls or pebbles are placed, the mineral particles being ground and the amalgam recovered. The rejected sulphides, carrying perhaps \$1,000 per ton in gold, are briquetted with water glass and charged to the blast furnace, when smelting cyanide precipitate or byproducts.

The increased proportion of the amalgam recovered from the skimmings during the second period under review is a natural result of the increased percentage of sulphide particles in the ore. The item is, of course, more properly a credit to the earlier ones, presumably largely to those for the battery and first-row. The tendency of fine sulphides to collect in the slight depressions in the surface of an amalgamated plate is well illustrated at the regrinding mill, where it is necessary to wash such a film from the surface several times each day.

It is perhaps not customary to report inquiries which have failed of definite conclusions. The following notes of such an investigation are not without interest, despite the lack of decisive result. Records of all six mills were tabulated over a period of one year, in an effort to determine

what relation, if any, existed between the amount of quicksilver fed to the batteries, the amount of bullion recovered, and the grade of the ore crushed. As was to be expected, individual determinations varied considerably from the average, but for four of the mills, and these fortunately including those crushing ore of the two extremes in value, the points coincide reasonably well with a curve represented by the equation

$$T = \frac{8.5 B - H}{H}$$

when

T = Value of ore (dollars per ton).

H = Troy ounces of quicksilver fed to batteries.

B = Troy ounces crude bullion (\$16 per ounce) recovered.

Of the two remaining mills, one was crushing surface ores with comparatively coarse particles of free gold. This, not unreasonably, showed values for H about 10 per cent. below the values indicated by the equation. The sixth mill gave results nearly 20 per cent. higher than might have been expected. This mill, on account of an insufficient supply of water, crushed nearly 10 per cent. less ore than the average and it is conceivable that the longer retention of the material in the battery caused undue flouring of the quicksilver. It is more probable, however, that the explanation is to be found in the personal equation of the millman in charge.

The crushing units are not in conformity with modern ideas, and there is no doubt that, were the plants to be built anew, radical changes in design would be made. The Homestake, unlike many younger mines, has developed gradually from comparatively small beginnings. The mills represent the growth of 30 years. Built before the cyanide process was known, they then represented advanced milling practice. It must be conceded that the stamp duty of $4\frac{1}{4}$ tons, with 80 per cent. of the tailing passing a 100-mesh screen and 60 per cent. passing a 200-mesh screen, is fairly good even when compared with the results reported from many newer installations.

We find that heavier stamps crush more rock, with little or no change in the sizing of the tailing unless more open screens are used; in which case, even without increasing the falling weight, tonnage may be gained at the sacrifice of sizing. To us it appears that compactness of plant, with the saving both in first cost and, later, in labor and supervision, constitutes the leading claim of the heavy stamp for preferment. Indeed, if the gold metallurgist is ready to dispense with amalgamation—and we are by no means prepared to do this—it is more than probable that he will do well to investigate the crushing practice of modern copper mills before he commits himself to the heavy gravity stamp. Alaska, at the moment, holds more of interest than does Africa.

It may be pertinent to note that I have never encountered a Home-

stake mill tailing, no matter how far advanced in secondary treatment, from which some free gold could not be recovered by laboratory amalgamation tests.

Sundry operating and cost data of stamp milling are appended:

Stamp Mills: Analysis of Lost Time

NOTE.—The time devoted to dressing plates, about 10 battery-minutes daily, is in part included, as whenever possible the work on the battery is done during this period of plate dressing. Data of 660 stamps for 59 consecutive days.

Reason for Loss of Time.	Battery Time Lost.	
	Hr. Min.	
Installing new screens.....	69 : 20	
Installing new shoes, dies, heads.....	194 : 40	
Installing new guides, guide castings.....	10 : 35	
Installing new mortars.....	63 : 55	
Installing new tappets, cams.....	20 : 00	
Installing new cam shafts.....	6 : 25	
Installing other equipment.....	37 : 00	401 : 55
Setting tappets.....	685 : 10	
Pullouts of stems.....	122 : 20	
Broken stems.....	343 : 05	
Broken shoes and dies.....	15 : 10	
Loose cams.....	39 : 50	
Miscellaneous repairs.....	120 : 05	1,325 : 40
Motor trouble.....	253 : 20	
Drive troubles.....	5 : 20	258 : 40
Total lost time.....		1,986 : 15
Total battery hours.....		186,912
Percentage of total time lost.....		1.07

Cost of Stamp Milling

	Stamp- ing	Amalgamating		Total
		Normal Charges	Rebuilding Plate House	
Operating labor.....	\$0.0895	\$0.0164		\$0.1059
Other labor.....	0.0033		\$0.0064	0.0097
Power.....	0.0457			0.0457
Machinery.....	0.0628			0.0628
Water.....	0.0315			0.0315
Sundry supplies.....	0.0097	{ 0.0036 quicksilver 0.0042 silver plating 0.0003 miscellaneous	{ 0.0059 silver plating 0.0018 lumber	0.0255
Total.....	\$0.2425	\$0.0245	\$0.0141	\$0.2811

Power at \$30 per horsepower year; quicksilver at 50c. per pound; castings, 2½c. per pound; mill labor, \$2.97 per 8-hr. shift.

REGRINDING

Operations at the regrinding plant offer little of interest, the practice conforming closely to the usual methods.

The feed to the tube mills is already so fine (only 25 per cent. remains on a 50-mesh screen) that the efficiency of the mills is low. The critical size of sand delivered to the sand plants is about 100 mesh for unoxidized ore, and 80 mesh for oxidized ore. It is advantageous to operate the mills in closed circuit, but it is difficult to do this without permitting sulphide particles which have been sufficiently reduced in size, to remain in the circuit. By introducing a double baffle or trough classifier, containing a hydraulic device, into the slime-overflow end of a Dorr classifier this difficulty has been in a measure overcome, and two of the mills are at present using this system. The accompanying table of tube-mill data has been compiled from earlier records, when none of the tube-mill discharge was returned for further grinding.

Each mill discharges its tailing over an amalgam table, and about \$0.40 is recovered on these per ton ground.

Regrinding Plant Data

Mill Number	1	2	3	4
Manufacturer.....	Denver Engr. Works.	Allis-Chalmers Co.	Allis-Chalmers Co.	Hardinge Conical Mill Co.
Dimensions, feet.....	5 by 14	5 by 18	5 by 18	6 by 6
Speed, rev. per min.....	27.0	28.5	28.0	26.5
Actual horsepower (motor input)...	32.0	42.0	42.0	27.0
Fed from.....	Dewatering cone.	Dorr classifier	Dorr classifier	Dewatering cone.
Tons fed per day.....	98.0	138.0	133.0	93.0
Pebbles per ton.....	1.40	1.24	1.38	1.71
Tons to pass 100 mesh.....	41.0	52.0	49.0	34.0
Tons to pass 100 mesh per horsepower.	1.28	1.24	1.17	1.26

Regrinding Plant: Operating Costs for 1914

	Cost per Ton Fed to Tube Mills	Cost per Ton Reduced to Pass 100-mesh Sieve
Labor.....	\$0.0496	\$0.1289
Pebbles and liners.....	0.0259	0.0673
Renewals, mills and cones.....	0.0074	0.0193
Machine-shop service.....	0.0037	0.0096
Silver plating.....	0.0077	0.0200
Sundries.....	0.0029	0.0075
Power.....	0.0292	0.0759
Total.....	\$0.1264	\$0.3285

Tons fed to mills, 152,106. Tons ground to pass 100-mesh sieve, 58,605. Cost of regrinding, per ton crushed in stamp mills, \$0.0121.

SAND TREATMENT

The sand is leached with cyanide solution in vats 44 ft. in diameter by 9 ft. deep, holding 610 tons of sand each. The operations are distinguished by unusual care in the preparation of the material for extraction, rather than in the extraction itself.

Classification and aeration are the two essentials to successful work. The latter, necessary to overcome the tendency of the ferrous compounds in the ore to remove the vital oxygen from the solutions, is achieved by forcing, at intervals, air under slight pressure into the false bottom, below the filter canvas. The sand charge is suitably drained before this air is applied; the pressure is so adjusted that the air is forced into the charge, yet is so low that the column of sand is nowhere broken or disturbed. To secure this immunity thorough classification is essential, and this has always been recognized as a matter of first importance.

Four batteries of cones, each fed from the discharge of the preceding set, the first three sets acting by gravity alone, the last assisted by a hydraulic connection, deliver a very clean sand to the vats. As a further precaution, the vat is filled with water before sand is turned into it, so that a constant overflow is maintained during filling and a final separation of slime particles added to the sum of the cone separations. As a matter of fact, the sand is so clean when it enters the vat that this final step is not one of strict necessity, but we have found that a charge filled in this manner is in a less compacted condition, presumably easier to leach and certainly easier to discharge.

The action of the air is interesting. Laboratory tests indicate that, on an average, about 75 cu. ft. of oxygen is absorbed by a ton of ore, before its reducing action is corrected. If this air is not supplied from some extraneous source, the solutions are vitiated and extraction ceases. When air is applied and is followed by a water wash, calcium thiosulphate appears in the effluent solutions; when the air is followed by cyanide, the effluents contain sulphocyanides and free cyanide appears only after some time. After a limited time of leaching, the extractive power of the solution decreases and further aeration of the charge is necessary. Each aeration is attended by the formation of some acid and the consequent destruction of some lime and cyanide. After treatment, the sand shows no sign of oxidation; neither analytical nor microscopic examination can detect differences between charge and residue.

Lime, crushed in a one-stamp mill to pass a 7-mesh screen, is added to the pulp stream as it flows to the vat. Regulation is by weighing a prescribed amount of lime into the automatic stamp feeder at 2-hr. intervals. The choice of screen is determined by the rate at which the lime slakes, the intent being to exhaust the particles only when treatment is completed. Formerly small quantities of lime were added to the top of charges dur-

ing the drainage periods, to supplement that fed with the sand, but this is no longer considered to be necessary and is not done except in emergencies.

It had long been recognized that a high protective alkalinity was detrimental, but until within the past two years it had been tacitly accepted that it would be impossible to dispense entirely—or practically so—with protective alkali. Eventually this was done; the quantity of lime added was reduced to the minimum necessary to correct the acidity of the ore; extractions improved and consumption of cyanide was materially reduced.

The gratifying results are in a measure due to another change in procedure, initiated at about the same time. Reference to the treatment chart will show that solutions of two strengths are used; that stronger in cyanide is used earliest in the treatment of a charge. When these solutions appear as effluents, one portion is precipitated, the other brought to full working strength by adding cyanide and thereafter returned to the extraction of another charge of sand.

This custom is by no means unusual, yet wherever it is used the procedure seems to be to maintain the stronger solution at a fixed content of cyanide and to allow the weaker to vary in strength according as the varying losses in treatment may determine. This is wrong in principle. The critical strength is that below which the weaker solution will no longer dissolve its quota of gold, or, if dissolving it, will no longer freely yield it to precipitation. If the strength is greater than this, we may confidently expect some cyanide to be destroyed without giving compensatory service. In such circumstances, if the maximum working strength is reduced, the weaker solution will be automatically restored to the lowest economic strength; if it falls below this, a temporary increase in the maximum strength will restore it. In other words, cyanide should be added to the *strong* solution in accord with the determinations of strength made on the *weak* solutions. Thus, operating with a strong solution of variable strength and a weak solution also variable, but never far from its effective limit, it may be anticipated that little cyanide will be needlessly expended.

The following definitions are required in connection with the treatment chart (Fig. 3).

Leaching Rate is the fall of the surface of solution in inches per hour when the charge is completely covered.

Alkalinity is expressed as the number of cubic centimeters of tenth normal acid required to neutralize 100 c.c. of the solution tested.

1 unit = 0.0028 per cent. CaO.

Protective alkalinity = $(0.056a - 11.4n)$ (in terms of pounds CaO per ton) when a = alkalinity and n = per cent. NaCN.

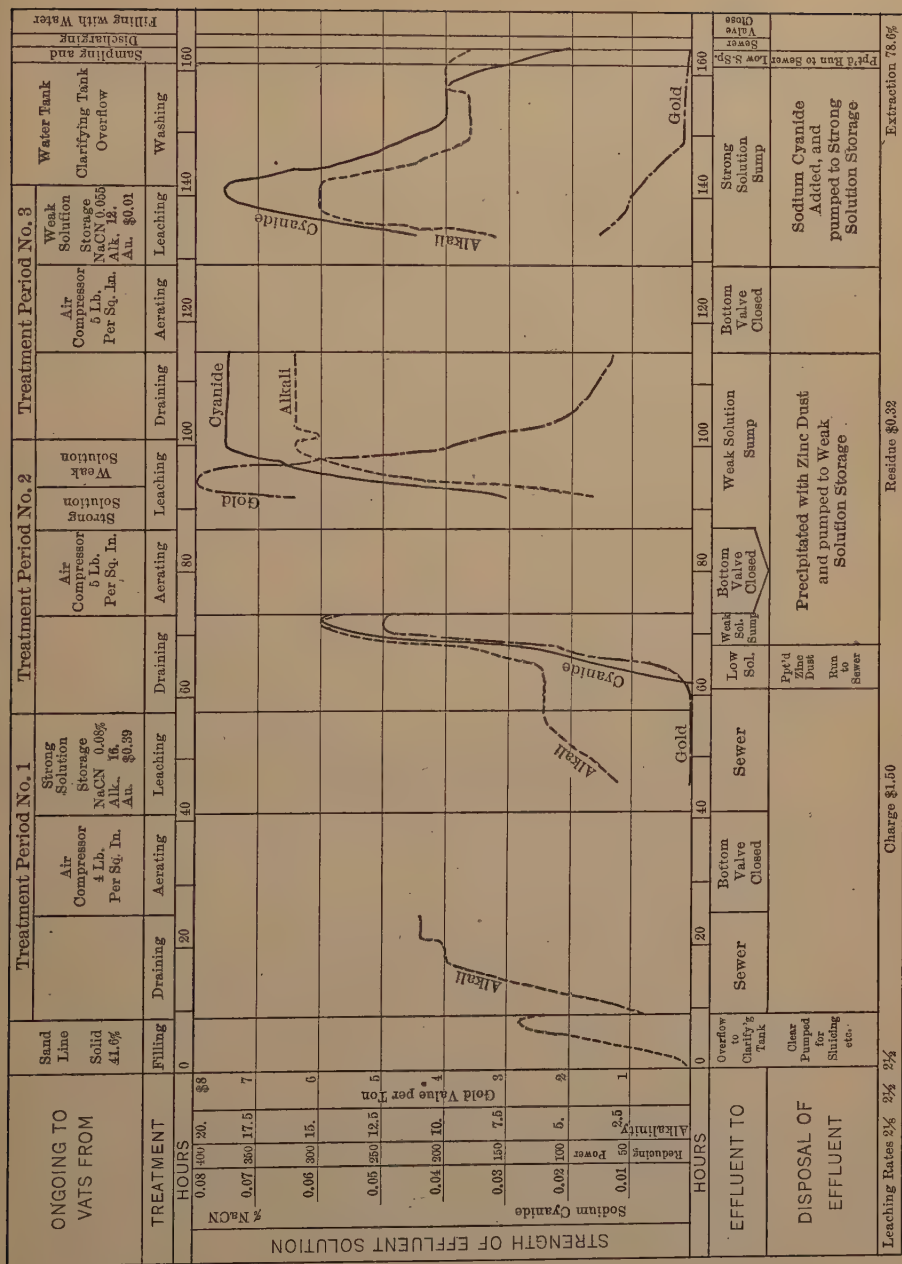


FIG. 3.—CHART SHOWING TREATMENT AT SAND PLANT No. 1. Data secured in March, 1914.

No. 1 Sand Plant: Operating Costs for 1914

Tons treated, 598,043

	Operating Labor	Other Labor	Power	Chemicals	Misc.	Total
Superintendence.....	\$0.0081	\$0.0081
Assaying.....	0.0033	\$0.0004	\$0.0006	0.0043
Transportation.....	0.0010	0.0001	0.0011
Neutralization.....	0.0034	\$0.0002	0.0093	0.0129
Classification.....	0.0086	\$0.0002	0.0007	0.0008	0.0103
Treatment.....	0.0293	0.0011	0.0054	0.0541	0.0021	0.0920
Precipitating and pump- ing solutions.	0.0055	0.0005	0.0005	0.0046	0.0006	0.0117
Heating.....	0.0007	0.0041	0.0048
Refining.....	0.0015	0.0022	0.0037
Miscellanies.....	0.0019	0.0009	0.0005	0.0024	0.0057
Repairs ^a	0.0121	0.0072	0.0193
Power from H. M. Co.....	0.0033	0.0033
Total.....	\$0.0632	\$0.0148	\$0.0106	\$0.0684	\$0.0202	\$0.1772

^a Includes only larger items of repair work, necessitating the labor of a crew of carpenters.

Unit Consumptions per Ton of Sand Treated

Sodium cyanide.....	0.257 lb.
Zinc dust.....	0.061 lb.
Lime.....	2.48 lb.
Power.....	1.09 kw-hr.

SLIME TREATMENT

The slime plant is equipped with 30 Merrill presses, each of 90 frames, 4 ft. by 6 ft. by 4 in. One press holds 26 tons of slime, and has a capacity of about 70 tons a day.

The entire treatment is given in the presses, which successfully meet the conditions for which they were designed, holding a tight and homogeneous cake during aeration periods (which impose a most exacting requirement on the machines) and discharging readily, without requiring manual labor. In this connection it may be mentioned that all the operations incident to the filling, aerating, leaching, washing, and discharging of 30 presses—totaling about 75 charges daily—are conducted by three men on each shift. The actual time required by one of these men in the performance of every step in the treatment of a charge is slightly less than 15 min.

Treatment is in principle identical with that of the sand, periods of aeration and treatment alternating with sufficient frequency to maintain the activity of the solutions. The alkalinity is carried at a very low point, and the working solutions are of low cyanide strength, as reference to the

charts will show. Within the restrictions imposed by practice, there seems to be no solution so weak that it will not freely extract gold from the aerated slime. The problem seems to demand the passing of a definite volume of solution through the slime cake, rather than treating with solution of a definite strength or for a definite time. So long as this amount of solution is passed, the extraction is accomplished. Thus the treatment time is a function of the leaching rate.

The minimum working strength is determined by our ability to precipitate the effluent solution, and in developing this minimum much of the treatment is given with very dilute solutions, with the result that the consumption of cyanide has been reduced to an extremely low figure. For instance, a 10-day period, concluding as this is written, gives the following:

Sodium Cyanide (128 Per Cent.) Consumed per Ton of Slime Treated

In treatment.....	0.127
Added for precipitation.....	0.008
	<hr/>
Total.....	0.135

The precipitation of the "low solution" (first and final effluents, low in both gold and cyanide) is difficult; doubly so on account of the low alkalinity; and after trial of many expedients to assist it we are not able to improve upon Carter's old system of a drip of strong solution.

This, usually added with the feed of zinc dust at the rate of 1 lb. per hour (or per 25 tons of solution precipitated) constitutes from 5 to 10 per cent. of the cyanide used. An excessive amount of zinc dust is also necessary to insure clean precipitation of these solutions, and this entails further expense, not merely in the first cost of the zinc, but in the refining of the larger bulk of low-grade precipitate. Combined, these costs consume about 25 per cent. of the profit won through the reduction of cyanide consumption, a satisfactory enough result in a plant where treatment methods have been so well established as is the case here, even though the impression may be conveyed that one branch of the work has lost efficiency.

Some assistance in this precipitation difficulty is received through the application of this barren low solution as a preliminary wash.

Treatment charts (Figs. 4 and 5) give full details of the treatment of typical fast- and slow-leaching charges. The leaching rate is stated as the number of seconds required to fill a 5-gal. bucket with effluent solution, the press being full of slime and the ongoing solution being applied at 30 lb. pressure.

Double cloths are used on the treatment presses, the lighter one being in contact with the frame. This is of unbleached muslin twill, and is in

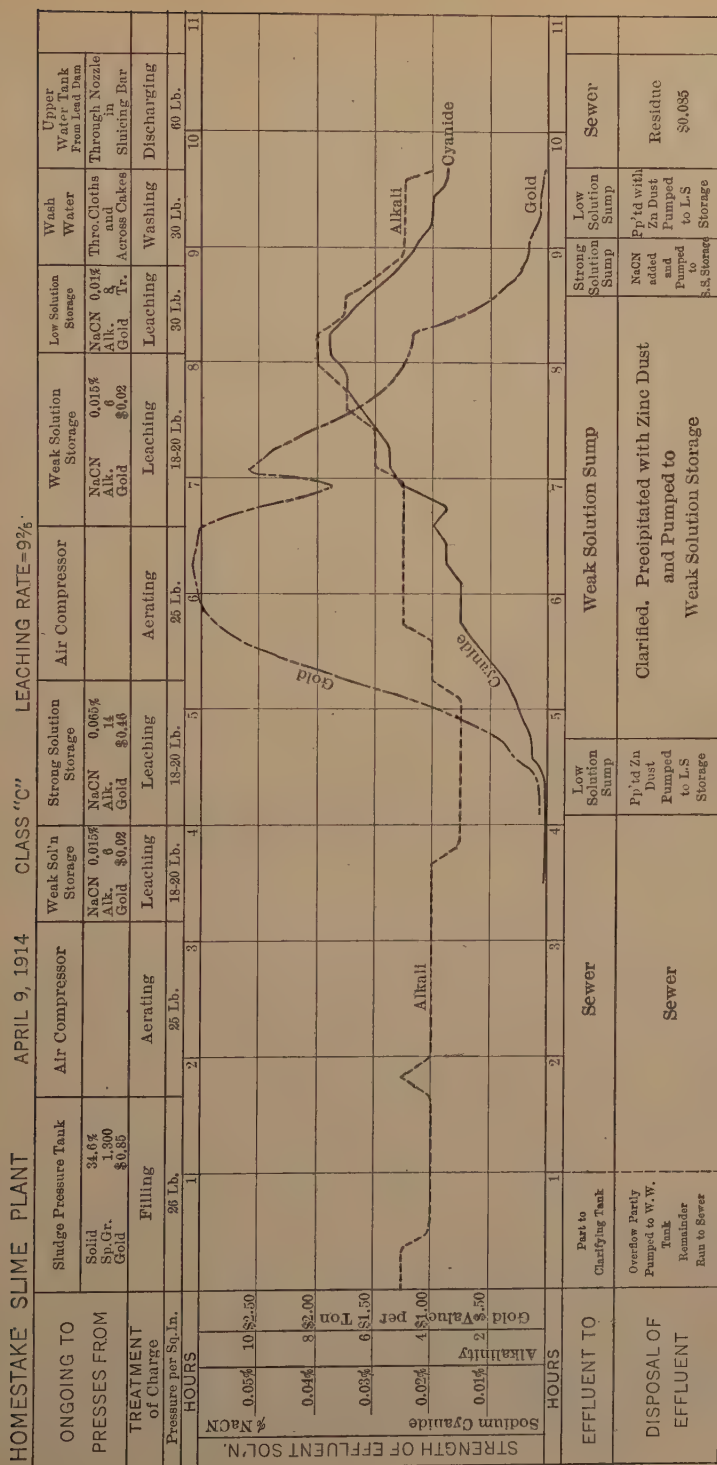


FIG. 5.—TREATMENT OF SLOW-LEACHING CHARGE, HOMESTAKE SLIME PLANT.

turn covered with a cloth of No. 10 canvas duck. Joints are made with Thermalite paint.

During the early days of the plant, when the equipment was not equal in capacity to the tonnage of available slime, a suit of cloths was used for as long as 30 months, it being preferable to lose slime rather than to stop to repair leaks. It should be remembered that the arrangement of ports and channels in the presses is such that treatment may be given in either direction, so that a single leak would in no wise affect treatment, but would merely entail the wasting from the plant of a certain amount of untreated slime. At this time, as much as 500 tons was wasted in a month. As the plant has been brought to full capacity, the cloths have been renewed, first after 18 months' service, now after 16 months; the leakage loss on the latter basis has been brought to an average of less than 40 tons monthly over an entire year, with a minimum of 8 tons in a single month. This in treating a total monthly tonnage of more than 58,000.

The "flow-gravity" system of tonnage measurements, elaborated by W. J. Sharwood (*Mining Magazine*, vol. i, No. 3, p. 226 (November, 1909), finds an interesting application in the estimate of slime treated at this plant. Here, even were it possible to determine the weight of the slime cakes, it would still be impossible to know the tonnage treated, since the amount of slime retained in a press after sluicing cannot be absolutely known. Certain classes of slime are more difficultly discharged than others; the stoppage of a single sluicing nozzle leaves 1.1 per cent. of the press unsluiced. Frequent testing of sluicing bars and examination of presses after sluicing protect us from loss of efficiency from this cause, so that the sluicing is never less than 96 per cent. efficient, and is at present, in my judgment, close to 99 per cent.

To at once determine the net capacity of a press, the "flow-gravity" system is used in a large way, measuring the total tonnage received at the plant during a 24-hr. period. This method is simple, thorough, and convincing, and since it is not so generally known as its value warrants, the full report of such a test is given herewith.

Procedure.—Valves on the two "sludge tanks" receiving the incoming stream of pulp are closed. The depth of slime in each tank and the time are noted. The stream of pulp is now diverted to tank A, the discharge valve from which is closed, the presses being served with pulp delivered from tank B. When tank A is filled, the time and volume are noted, the valve from B is closed, the pulp turned to tank B and the discharge valve from tank A is opened. These alternations continue throughout the period assigned for measuring. Changing, which is at intervals of about 35 min., is not instantaneous, but the actual measurements cover 90 per cent. of the total time, sufficient to insure a correct average rate.

At half-hourly intervals the stream of pulp is cut by a mechanical sampler, a 5-gal. bucket being filled and the weight of pulp determined. These determinations are averaged. A specific gravity determination is made on the average sample of slime.

Estimate of tonnage treated between noon of Apr. 8, 1915, and noon on following day:

	Feet	Cubic Feet
North Sludge Tank:		
Total depth pulp to tank.....	132.667	
Deduct, lime water.....	1.078	
	<hr/>	
Balance, slime pulp.....	131.589	
131.589×518.85 (cu. ft. per foot of depth)....		68,274.95
South Sludge Tank:		
Total depth pulp to tank.....	125.334	
Deduct, lime water.....	1.003	
	<hr/>	
Balance, slime pulp.....	124.331	
124.331×514.42		63,958.35
		<hr/>
Total cubic feet for 22.05 measured hours.....		132,233.30
Estimated pulp for 24 hr.....		143,927.28
Correction for excess pulp in tanks at end of test.....		3,590.00
		<hr/>
Cubic feet of pulp run to presses during test.....		140,337 +

During the test, 39 determinations were made of the weight of 5 gal. of pulp. The average of these determinations gave the following data:

Percentage of solids in pulp.....	33.68
Weight of 1 cu. ft. of pulp.....	80.60 lb.
Solids in 1 cu. ft. of pulp.....	27.146 lb.

Total solids to presses during test = $140,337 \times 27.146 = 3,809,600$ lb. = 1,904.8 tons, which was divided among 73 presses, making the average charge 26.09 tons.

This scheme of tonnage measurement has been elaborated at our mills during the past eight years, and has found many useful applications. Two more of these may be cited. One, another application in a large way, was described in *Transactions of the Institution of Mining and Metallurgy*, vol. xxii, p. 201 (1912-13) from which I quote:

“At the start the sand vat was completely filled with water. A large tank (1246 cub. ft.) was arranged so that the entire overflow (equalling in volume the entering pulp) could be switched into it, and the time of filling the 1246 cub. ft. noted. This tank was filled 53 times at equal intervals during the 48 hours required to fill the sand vat, the total volume of pulp being thence calculated as 92,711 cub. ft. Meanwhile the pulp was systematically sampled by a mechanical cutter, a standardized bucket of $\frac{2}{3}$ cub. ft. capacity (5 U. S. gallons), being filled and weighed 116 times at equal intervals. The samples were all thrown upon a filter, to be dried and weighed later. The specific gravity of the dry solid was accurately determined. From these data the following computations were made:

- (b) Sp. gr. of ore 2.985. Average weight of 116 buckets [of 5 gal.] = 54.575 lb., or 81.862 lb. per cub. ft. pulp. Average sp. gr. of pulp = 1.31, whence 1 cub. ft. contains 29.062 lb. dry sand. $29.062 \text{ dry lb. sand per cub. ft.} \times 92,711 \text{ cub. ft.} \div 2000 = 1347.2 \text{ tons [sand in vat].}$
- (c) Total dried sand from 116 buckets . . . weighed 2261 lb., or 29.24 lb. per cub. ft. pulp. $29.24 \text{ lb. per cub. ft.} \times 92,711 \text{ cub. ft.} \div 2000 = 1355.4 \text{ tons.}$
- (a) Tonnage for a depth of 160 in., based on former accepted weight of cub. ft. boxes = 1344.0 tons.

Maximum variation = 11.4 tons = 0.85%."

Another convincing demonstration of its accuracy, under more difficult conditions, has recently been noted. A detail of stamp milling was under investigation. Tonnage measurements of the trial battery and of a standard battery were made by diverting the entire flow of a five-stamp battery into a 15-gal. container, noting the time required to fill the container and the weight of the impounded pulp. The time required to fill the container, under these conditions, is from 20 to 25 sec.; the water ratio is over 10:1. Although many repetitions carried conviction of the general accuracy of the observations, it was thought best to confirm the results by weighing all ore supplied to the batteries, this decision being based upon the double liability to a large percentage error on account of the brief sampling time and the dilute pulp. Accordingly, a scale was installed, with the result that the earlier determinations were confirmed within 2 per cent.

A statement of operating costs at the slime plant is appended. This plant was the pioneer of its type. It is interesting to note that the presses are now practically as at first designed, and that the process, in its essentials, is unchanged. Certain refinements in the press design have been dispensed with as unnecessary; a study of the operations has made possible some simplification and systematization, which have made it possible to reduce the costs each year.

Slime Plant Operating Costs for 1914

	Operating Labor	Other Labor	Power	Chemicals	Miscellaneous	Total
Superintendence.....	\$0.0093	\$0.0093
Assaying.....	0.0021	\$0.0005	\$0.0005	0.0031
Thickening.....	0.0033	0.0001	0.0034
Neutralization.....	0.0055	\$0.0004	0.0144	0.0203
Treatment.....	0.0342	\$0.0027	0.0152	0.0433	0.0119	0.1073
Precipitating and pumping solutions.....	0.0039	0.0006	0.0016	0.0096	0.0008	0.0165
Heating.....	0.0004	0.0013	0.0017
Refining.....	0.0026	0.0040	0.0066
Miscellaneous.....	0.0057	0.0009	0.0012	0.0010	0.0088
Repairs ^a	0.0042	0.0026	0.0068
Total Operating....	\$0.0670	\$0.0084	\$0.0184	\$0.0678	\$0.0222	\$0.1838

^a Minor repairs are carried as Miscellaneous; heavy items of repairs, necessitating the presence of carpenter's crew, are included here. Press cloths cost \$0.0100 per ton of slime treated.

Unit Consumptions per Ton of Slime Treated

Sodium cyanide.....	0.161 lb.
Zinc dust.....	0.118 lb.
Lime.....	3.84 lb.
Hydrochloric acid.....	0.393 lb., costing \$0.0093 per ton treated.
Power.....	1.15 kw-hr.

PRECIPITATION

The Merrill system of precipitation with zinc dust is used at all cyanide plants, as is the Merrill filter press of triangular section. This method has been fully discussed in the paper already cited, and the technique remains about as there described. It has been mentioned that the effect of the recent variations in treatment has been adverse to precipitation, since the tonnage of solution requiring precipitation has been increased at the same time that the content of solution in gold and cyanide has been reduced. Beyond the natural effects—increased zinc consumption and lower-grade precipitate—no difficulties have been encountered.

Double cloths are used on the frames, the outer one being taken off when the press is cleaned and replaced by the inner cloth, which is in turn replaced by a new cloth. The outer cloth is burned, the ash going to refining.

Experiments are still in progress to find the lightest and cheapest cloth that can be used for this purpose without unduly increasing the hazard of leakage.

All washing of cloths is eliminated by this system. Against the added cost of cloth may be set the advantage of security from leaks due to rotten or wrinkled cloths and the labor of washing soiled cloths.

REFINING

In its essentials, this remains as described in the earlier paper, except that after acid treatment of the lower-grade precipitate, the briquets are charged to the blast furnace instead of being fused in the cupel. The latter treatment is still the standard for precipitate of higher grade.

This change and the general scheme of treatment are shown in Fig. 6.

Acid treatment is a preliminary to furnace treatment of all precipitates. It may not be necessary, but in my judgment it is sound metallurgy, since it enhances security from stack losses and insures more fluid slags. Moreover, the decreased bulk of product to be smelted with lead decreases the time required for that operation, always more or less hazardous to the health of the workmen.

Oil is now used as a fuel in the cupel furnace, replacing wood. It is an ideal fuel for the purpose, and its use has resulted in an acceleration of nearly 50 per cent. in the rate of cupelling.

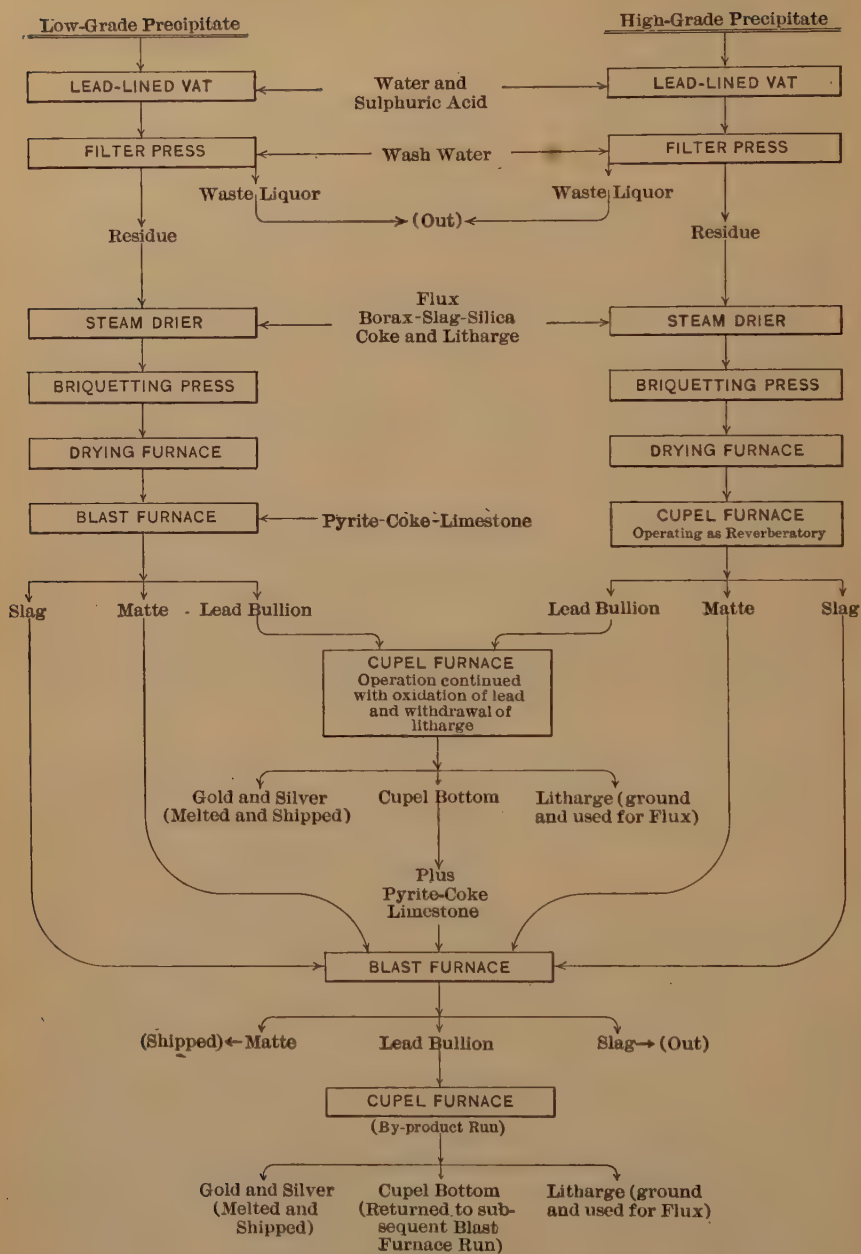


FIG. 6.—REFINING OF PRECIPITATE AT HOMESTAKE.

DISCUSSION

G. H. CLEVENGER, Palo Alto, Cal.—Since the precipitation of extremely dilute solutions as regards cyanide presents one of the greatest problems of zinc-dust precipitation, I should like to ask Mr. Clark the minimum concentration in cyanide of the solution from which it is possible to precipitate satisfactorily the gold with zinc dust in Homestake practice.

A. J. CLARK.—For a day's work about 0.015 per cent. That is precipitating a very low-grade solution, very low in alkalinity. It requires probably 0.2 lb. of zinc dust to the ton.

JOHN A. FULTON, Melones, Cal.—At the Melones mine, we have been working very low-grade rock for a number of years, the process being amalgamation and concentration up to the last year. The value of the ore increased so that we had to supplement the concentration with a sand and slime plant. Classification was a problem, since to install classifiers would mean tearing the mill all to pieces. We are therefore making use of a Wilfley table for this purpose. The amount of wash water is increased and a stream of fresh water is also run into the feed box at the end away from the intake; this gives a clear line of demarcation between the sand and the slime, enabling us to draw off sand at one end of the table and slime at the other.

Metallurgical Practice in the Witwatersrand District, South Africa

BY F. L. BOSQUI, PH. B., M. D., JOHANNESBURG, TRANSVAAL

(San Francisco Meeting, September, 1915)

INTRODUCTION

THE history of the development of gold metallurgy in South Africa is divisible into two periods: That preceding the introduction of the cyanide process on a commercial scale in 1890; and the 24 years intervening between that important event and the present time.

The period between the discovery of the banket reefs of the Witwatersrand and the year 1890 was one of phenomenally rapid progress in mining development, but of no special interest to the metallurgist. There were no strikingly original or distinctly local advances in the treatment of gold ores. Californian practice in stamp milling and amalgamation prevailed, in mills ranging in equipment between 5 and 50 stamps, operating within a wide range of efficiency, and under the trying conditions usually encountered in a new and isolated field. The somewhat incomplete records of gold output of that period show that in December, 1887, there were 20 producing companies in the Witwatersrand district, of which 10 had mills, the largest being of 20 stamps. The only official record of returns for May, 1887, was that of the Wemmer mine, whose modest achievement with a 5-stamp mill was 100 tons crushed, producing 887 oz. 3 dwt. of gold. This is interesting in view of the present-day output of this field, which in 1913 reached 8,430,998 oz. The rapid increase of production during this first period (May, 1887, to January, 1891) is shown by the following summary for the year 1890:¹

Total tons crushed.....	702,828
Number of stamps in operation.....	1,046
Average number of tons crushed per stamp per day.....	2.36
Yield of gold, ounces.....	494,817
Yield per ton, pennyweights.....	13.36

The stamp duty ranged between $1\frac{1}{2}$ and 4 tons per day, according to weight of stamp, fineness of screen, and operating skill available. The prevailing weight of stamp was about 900 lb., and the screen, 900 meshes to the square inch.

¹ *Annual Report of Johannesburg Chamber of Mines, 1889-90.*

Assuming that the recovery at that time was as high as 60 per cent., the recoverable gold lost in tailing from the ore treated during this first period, in spite of the precaution taken in some instances to impound this product, must have been very considerable. It is evident, therefore, in view of the colossal strides made by the industry at this stage, and the promise revealed for the future, that the advent of the cyanide process was most opportune.

The introduction of the new process was the direct result of trials conducted at the old Salisbury battery in 1890 by the Cassel Gold Extraction Co. of Glasgow, which controlled the MacArthur and Forrest patent rights. Its first application on a commercial scale was in the same year, in the treatment, by contract with the Gold Recovery Syndicate, of 10,000 tons of tailing on the Robinson Gold Mining Co.'s property, under the direction of G. A. Darling. In February, 1891, W. A. Caldecott started the cyanide works at the Sheba mine in the Barberton district; and a year later, Charles Butters erected a large plant at the Robinson mine to treat all the tailing from that property. From this time cyaniding was recognized as an indispensable adjunct to every mill. It was not, however, until 1894 that the direct treatment of battery slime became a success. The first decantation slime plant was erected in this year at the Crown Reef mine, under the direction of J. R. Williams, at that time chief metallurgist for the Rand Mines, Ltd., who evolved the process after several years of practical trials.

In the meantime, in 1894, the Siemens-Halske electrolytic method of gold precipitation was introduced at the Worcester mine, and was adopted by several other properties. But in spite of able advocates it never won general acceptance, owing to the extreme delicacy of the process, the formation of troublesome byproducts, and the high degree of skill required to maintain uniform efficiency; and after exhaustive comparative trials with zinc and electrolytic deposition at the Nourse Deep mine in 1898, it was finally abandoned in favor of zinc. Fortunately, a serious objection to zinc precipitation—namely, the difficulty of precipitating from the very dilute cyanide solutions used in slime treatment—was partly overcome at about this time by the use of acetate of lead as a “dip” for zinc shavings, which coated the zinc with metallic lead, forming a zinc-lead couple which greatly promoted galvanic action in the zinc boxes.

With the reversion to the use of zinc, Rand metallurgical practice may be said to have crystallized into a general scheme of milling and of separate sand and slime treatment by cyanide, which came to be looked upon as typically South African, having been, with the exception of stamp milling, evolved by local chemists and metallurgists. Even the introduction of tube mills in 1904 did not seriously modify the main features of the system. The tube mill became simply an accessory to the stamp. This survival of the broad principles of ore treatment, as developed by the

clever metallurgists whose services the industry was fortunate enough to secure at the beginning, has been attributed to a narrow disinclination to adopt new methods as applied in other parts of the world. This view, however, must be very considerably modified.

It is true that the metallurgists of the Rand were slow to accept the two most notable appliances introduced into the treatment of gold ores in the last 10 years—the tube mill and the vacuum filter. The former was introduced in Africa in 1904, after having been for some time a success in Australia; the latter in 1909, after four years of brilliant success in America and elsewhere. The Boer war undoubtedly retarded the introduction of tube milling, and it must be admitted that when once tried, and their merits proved, both innovations were very generally and enthusiastically adopted. The fact that such radical measures as “all sliming,” dry crushing, and numerous schemes for continuous treatment of battery pulp by agitation have not found favor on the Rand is evidence, not of lack of enterprise or open-mindedness, but of the proper estimation of unique local conditions, and the conviction that any system of treating the low-grade Rand ores must fulfill the essential requirements of “foolproof” simplicity in operation, low maintenance and treatment cost, and high efficiency.

The local peculiarities of the district have not always been understood by critics. The ore of the Rand is not only low grade (ranging in gold content between $4\frac{1}{2}$ and 9 dwt. per ton), but is simple in composition, containing as high as 70 per cent. of its gold in amalgamable form, and a small portion in pyrite, which is quite amenable to cyanide treatment. Obviously, an ore of this grade and character does not permit of elaborate appliances or refinements of treatment. Moreover, it must be remembered that unskilled labor is cheap on the Rand, while skilled labor and technical advice are expensive. Supplies are also high priced; and electric power is comparatively costly. The country, being flat or gently undulating, does not afford such favorable sites for reduction plants as where gravity can be taken advantage of in the flow of mill pulp. This disadvantage of site also means added expense for disposal of residue, which must be carefully impounded or stacked. It will be seen from the foregoing that the economic aspect of Rand metallurgy is a study of some complexity, involving a nicety of balance between technical and financial considerations, and requiring the utmost caution on the part of metallurgical and mechanical advisers. This will explain the retention of methods and apparatus which, to the stranger fresh from the exhilarating atmosphere of innovation, may seem crude and awkward, but which have survived the test of competition with rival processes, and have proved to be the best available for our special purpose. When the very high gold extraction on the Rand (94 to 97 per cent.) and the low reduction cost (3s. and less per ton in the newer plants) are considered, the scope for economic improve-

ments does not seem great. It may be mentioned here, however, that though, broadly speaking, the same general system of reduction is in use throughout the Rand, there are many variations in detail between old and new plants, both in mechanical arrangement and treatment, which will be dealt with in their proper place, under the following subdivisions:

Step in Treatment	Variations
I. <i>Ore Sorting and Breaking</i>	<ul style="list-style-type: none"> (a) Ore washed in trommels or in stationary chutes. (b) Ore sorted on slowly moving belts or on revolving tables. (c) Ore broken in two types of crusher, those in common use being the Blake swinging jaw type and the Gates gyratory.
II. <i>Stamp Milling</i>	<ul style="list-style-type: none"> (a) A range in weight of stamps from 1,150 to 2,000 lb. (b) Considerable variation in details of battery construction; also variation in mesh of battery screen dependent chiefly upon tube-mill facilities.
III. <i>Amalgamation</i>	<ul style="list-style-type: none"> (a) Use of battery plates, and so-called tube-mill plates for secondary amalgamation after tube milling. (b) No battery plates; all amalgamation done after tube milling. (c) Tube-mill plates, shaking and stationary. (d) Considerable variation in ratio of area of plates to tonnage.
IV. <i>Tube Milling</i>	<ul style="list-style-type: none"> (a) Tube mills of varying dimensions. (b) Varying ratio of number of tube mills to stamps. (c) Devices for thickening pulp and feeding tube mills. (d) Kinds of "liner" in use.
V. <i>Classification of Pulp</i>	<ul style="list-style-type: none"> (a) Coarse sand separated out and sent to tube mills. (b) Separation of sand from slime. (c) Classification in spitzkästen or hydraulic cones.
VI. <i>Treatment of Sand</i>	
1. <i>Collecting</i>	<ul style="list-style-type: none"> (a) Collecting tanks superimposed on treatment tanks, or on ground level, and fed by (1) movable hose, (2) Butters and Mein distributors. (b) Collecting tanks filled direct from classifiers or with solution-borne pulp from Caldecott sand-filter tables.
2. <i>Treatment</i>	<ul style="list-style-type: none"> (a) Sand dropped from superimposed collectors into treatment tanks. (b) Sand transferred by belt conveyor from collectors on ground level. (c) Sand collected and treated in the same tank from Caldecott sand-filter tables. (d) Sand residue trammed to dump in trucks. (e) Sand sent to dump by aerial bucket conveyor.
VII. <i>Treatment of Slime</i>	<ul style="list-style-type: none"> (a) By circulation with pumps and decantation. (b) By agitation in Pachuca tanks, followed by filtration with Butters vacuum filter.
VIII. <i>Gold Precipitation</i>	<ul style="list-style-type: none"> (a) On zinc shavings. (b) On zinc dust in Merrill presses.
IX. <i>Smelting Methods</i>	<ul style="list-style-type: none"> (a) In Tavener lead furnaces, with cupellation. (b) Direct smelting in pots.

SORTING AND BREAKING ORE

The term "banket" is the Dutch word for almond rock, or almond "candy," as we should say in America; and was applied by the Boers to the conglomerate ore of the district which, with its white pebbles incased in the darker matrix, bears a resemblance to that sweetmeat. The object of sorting, which was first introduced in 1892, is to separate this banket ore from the comparatively barren quartzites, or waste. The care with which sorting is done and the proportion of ore rejected as waste (from 14 to 20 per cent.) vary at the different mines, according to facilities, occurrence of orebodies, and a number of economic considerations.

Ore sorting on the Rand has been the subject of a good deal of discussion. It has been suggested that all sorting should be done underground, while authorities have differed as to what proportion should be done underground and what on the surface. A kind of crude preliminary sorting is now done below the surface, consisting in hand breaking of pieces of ore too large for convenient handling. Obviously, the technique of ore sorting is a matter to be dealt with independently on each mine. The present practice is to sort above ground, at a central sorting and breaking station, where both operations are usually carried on under the same roof. These stations are of great variety, but the general principle is the same in all.

The ore, on reaching the surface, is divided into two products by means of grizzlies, which take it direct from the skips in the headgear; the "fines," which it is impossible to sort economically, go direct to the mill bin; the coarser product is then delivered either (1) to grizzlies or (2) to trommels, where it is washed and screened preparatory to hand sorting. Or, in older plants, the grizzly and trommel are omitted, and washing is done by means of sprays on the same belt or table on which sorting is performed. Washing the ore is necessary in order to enable sorters to distinguish between banket and waste. It is now recognized that this operation cannot be thoroughly performed except in washing and screening trommels, and the latest plants are all equipped with this device. From the trommel or grizzly the washed ore is delivered to either (1) a revolving table or (2) a traveling belt. The former is annular in shape, from 18 to 30 ft. outside diameter, the horizontal rim or working surface not exceeding 42 in. in width. About 3 ft. of space is allowed for each sorter—young native lads being chiefly employed. The ore makes nearly a complete circuit and is finally removed by a plow, whence it passes to the breakers. The sorted waste is either dropped below into a bin, or on to the lower deck of a double-decked table, whence it is removed by a plow, and thence to the waste dump.

The advantage of the sorting table is that it permits of a compact plant, and so simplifies supervision; but the belt has now almost entirely

superseded the table, its advantages being that it is cheaper to install and operate and that it elevates and conveys the ore as well as provides a sorting surface. These belts vary in width between 30 and 40 in., operate at an angle of about 10° , and are provided with a special form of idler to give a concave surface for draining and preventing spill. They are made either of canvas, or of canvas and rubber. Their capacity is between 50 and 100 tons per hour, according to width.

The waste from the sorting belt is dropped either into (1) a bin, for direct transfer to dump, or (2) on to waste belts, arranged in a variety of ways. Or the returning portion of the continuous sorting belt may be made to deliver the waste into trucks, bins, or a cross belt to dump. A considerable problem in sorting stations is the disposal of the "washings" or muddy water carrying grit and slime which flows from trommels, grizzlies, or the sorting belt, according to the method of washing adopted. The following methods are in common use: (1) these washings are settled in special tanks or pits, and periodically shoveled out and mixed with the ore going to the mill bins; (2) they are delivered direct to centrifugal pumps, and discharged into the mill-pulp stream; (3) or they are subjected to screening in secondary trommels, the washed oversize passing to a fines belt, which joins the main conveying system, and so direct to the mill bins, the escaping water and fine material going to centrifugal pumps and thence to the mill-pulp stream.

The imperfect arrangements for washing ore and for disposal of these washings in many sorting stations on the Rand, and the crowded, patched-up appearance of the majority of these stations, lead to the conclusion that this stage in the metallurgy of Rand ores has been rather neglected in the past. One wonders why this step in reduction should not have crystallized long ago into some good, generally accepted scheme—instead of which we find the greatest variety of ill-designed, crowded, messy plants, seemingly in the first crude stage of development. It is only in a few of the newest mills that the design of sorting and breaking stations has received sufficient attention. We find these well lighted and spacious; the usual spill and mess prevented by the use of proper washing trommels, and neat arrangements provided for the collection of washings and their direction into suitable pump pits.

In the next plant to be built by the Rand Mines, Ltd., it is expected that the sorting and breaking station will embody all the latest improvements and facilities (Fig. 1). The bad practice of putting fines and slimy ore in the mill bins will be obviated. By means of the secondary washing trommels only the fines that will not pass a screen of 9 holes to the square inch will be sent to the mill bins; all finer material will be diverted to the tube mills. These washings should properly not join the mill stream direct, for the reason that, as the crusher station runs only 8 hr. in the 24, this large accession of water during this period presents a serious

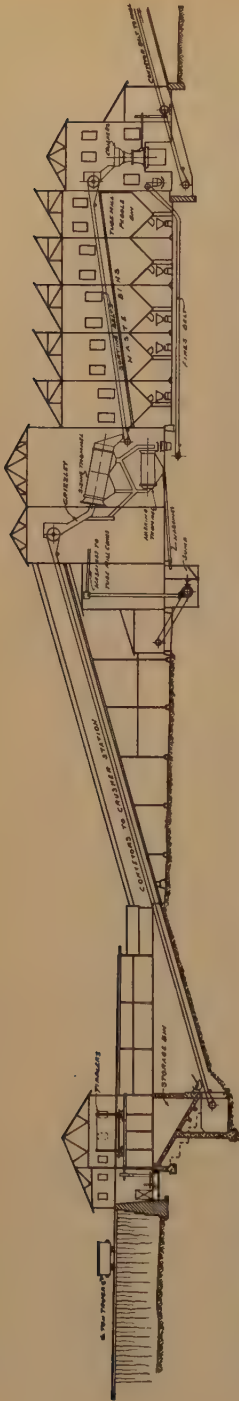


FIG. 1.—ELEVATION OF BREAKING AND SORTING STATION, PROPOSED NEW REDUCTION WORKS, RAND MINES, LTD.

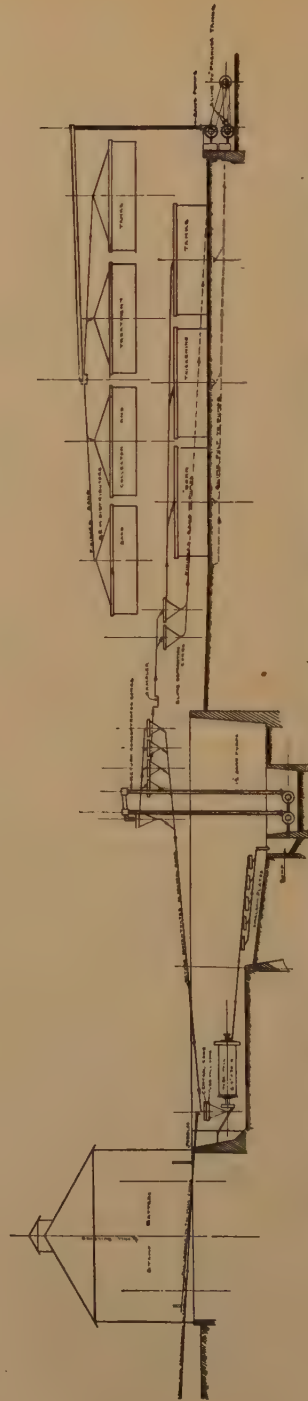


FIG. 5.—DIAGRAM OF CLASSIFICATION SCHEME, PROPOSED NEW REDUCTION WORKS, RAND MINES, LTD.

detriment to uniform classification. It is important, therefore, that it should be specially treated before reaching the mill. For this new mill a continuously operating special plant has been devised to deal with this troublesome product. It will run into suitable classifiers, the underflow of which will go to the tube mills; the bulk of the water and slime will be diverted to settlers, the last of the series representing the reservoir supplying the washing trommels. Thus the mill will be protected from irregular and large accessions of water, so undesirable where proper classification is recognized as of vital importance.

The sorted ore is delivered to breakers, of which there are two types in common use, the Blake-Marsden jaw type with its variations, and the gyratory type, of which the Gates is generally used. The gyratory breaker is the type now accepted by the Rand Mines and Consolidated Goldfields groups, and is considered more efficient and cheaper to operate where large tonnages are handled; the jaw crusher, however, is a simpler machine, of relatively less weight and requires less vertical height. The gyratory crusher is particularly well adapted to the finer breaking, while for coarse preliminary breaking, before sorting, where the production of fines is objectionable, the jaw type is preferred.

There is still a divergence of opinion as to the extent to which breaking before sorting should be carried. In some mines breakers are placed underground, with the jaws set to 4 to 8 in. opening, to reduce the ore to suitable size for handling in the skips and for sorting; more often these breakers adjoin the headgear. Where they are omitted altogether, it is in recognition of the theory that it is better to send a high percentage of waste to the mill than reduce the waste to smaller pieces, and so improve the accuracy of sorting while increasing the aggregate gold content of the waste dump. It will be seen from the foregoing that the practice in dealing with the ore before it goes to the mill is exceedingly variable. Within the last four or five years, however, the technique of sorting and breaking has greatly improved, and there is now apparent, among the groups erecting new plants, a disposition to collaborate in an effort to bring about more uniformity in practice and mechanical arrangement.

STAMP MILLING

The gravitation stamp has retained its place on the Rand as the simplest and most effective appliance for crushing ore. Until the recent introduction of the heavy single-stamp unit, the general features of ore crushing have undergone no very radical change since the introduction of Californian mill practice over 20 years ago. The most marked departure from the typical 900-lb. stamp of those days is in the progressive increase of weight. This interesting evolution from the light to the heavy stamp has probably now reached its limit in the adoption of 2,000-lb. stamps in

mills erected within the last few years. Changes in detail of battery construction have been necessitated by this progressive change in weight; exigencies of high stamp duty, requiring larger bin capacity, have also brought about modifications in constructional detail.

It would be impossible in the scope of this paper to deal with the many variations in stamp-battery construction to be found on the Rand. I shall consider only a few of the more important stages in progress, which have led to the introduction from America of the single-stamp unit, or Nissen stamp.

The first step in advance was the adoption by the Robinson Gold Mining Co. and others of 1,250-lb. stamps. This was the maximum until 1907, when a distinct impetus was given to a consideration of heavy stamps by the results of a series of trials conducted by the Consolidated Goldfields group under the direction of W. A. Caldecott. It was obviously desirable to ascertain the highest duty attainable from as heavy a dropping weight as could be sustained by an improved type of mortar box and concrete mortar block. It was apparent that any increase of duty would mean a corresponding decrease in number of units necessary, and, in consequence, lower initial and operating cost.

The Goldfields trials led to the installation of three large batteries of 1,550-lb. stamps, and more recent mills have adopted heavier weights up to 2,000 lb. Gradually timber-pile foundations were abandoned in favor of reinforced concrete, with or without the interposed anvil block. The stem and cam-shaft breakage attributed to this cast-iron base, and the higher cost of batteries using the anvil block, led to its abandonment in most of the more recent mills, although it still has its advocates. The accepted practice is now to bolt the mortar box direct to the concrete block by means of renewable bolts so arranged that they can be tightened while the battery is in operation. A cushion of hair felt or of sheet rubber, about $\frac{3}{8}$ in. thick, is placed under the mortar box.

A notable departure from the standard type of battery construction was the case of the City Deep, Ltd., which in 1909 adopted reinforced concrete piers instead of king posts, and separate bearings between the cams, for a new battery of 2,000-lb. stamps. This design was intended to obviate cam-shaft breakage by affording greater bearing surface, and to insure better alignment, in view of the greater strain imposed upon the cam shafts by the heavy stamps. At the City Deep the concrete piers built up from the 10-stamp block support a single rigid steel casting, into which all the various parts of the usual battery superstructure are consolidated. The main feature of this casting is the heavy fish-bellied girders, with eight upward projections or fingers, terminating in cup-shaped bearings for the cam shaft. Each 10-stamp shaft is thus given 11 bearings instead of the three usually provided. Theoretically, the innovation was sound, and ingeniously worked out. In practice, how-

ever, complications developed. It was found that a true alignment of bearing surface was practically impossible to maintain, without most troublesome and constant adjustment of the bearings themselves. Moreover, as the advantages of such a design were obviously dependent upon the integrity of the single ponderous casting, it was expected that this rigid frame would resist the enormous strain of continuous vibration. Unfortunately this casting in the course of time weakened, and developed fractures, requiring reinforcement and patching.

Following the City Deep installation, the old standard design of battery was reverted to and has been, with one exception, retained. In mills erected within the last five years we find the weight of stamps ranging between 1,550 lb., which some authorities reckon to be the conservative limit for combination of five units in a single mortar box, and 2,000 lb., which many regard as excessive weight, from the point of view of efficiency, convenience, and cost of maintenance.

The introduction of the tube mill in 1904 had much to do with the development of heavy stamps. Until that time fine crushing in the battery was necessary, in order to secure good recovery by amalgamation and cyaniding; but with the advent of the tube mill it became possible to use screens of coarser mesh, the limiting factor being the coarseness permissible for good amalgamation. It was obvious, therefore, that if battery amalgamation plates could be dispensed with, the only factor limiting coarse crushing would be the size of particles permissible in the tube-mill circuit. The removal of battery plates was therefore the next step toward attaining maximum stamp duties; and now in all the newest mills amalgamation is only carried on after tube milling. We thus find the function and scope of the stamp quite altered; it is now recognized as a crushing device pure and simple, hampered in capacity by tube-mill limitations only; so that, beginning with duties of a few tons from 900-lb. stamps when screens of 900 mesh were necessary, we are now obtaining duties of approximately 30 tons, with 9-mesh screens (0.27 in. aperture) from 2,000-lb. stamps. There is still, however, a considerable range in weight and duty of stamps on the Rand. Many older plants still retain battery plates, and are not provided with the same tube-mill facilities as newer mills. This is because capital expenditure on old plants is not always warranted by the life of the mine and the financial position of the company; this must be borne in mind, as otherwise the variable data given in the following tabulation, from the mills controlled by the Rand Mines, Ltd., may appear inexplicable.

Mills on the properties of other groups are obtaining stamp duties as high as 20 tons per day, from 2,000-lb. stamps, with 9-mesh and 4-mesh screens; but the highest duty yet recorded is that of the 2,000-lb. Nissen stamps at Modderfontein B., 29 tons through 9-mesh screen.

Averages for the Year 1913, Rand Mines, Ltd.

Company	No. of Stamps		No. of Tubes		Average Dropping Weight of Stamp	Duty of Stamp per 24 hr.	Screens Used
	Erected	In Operation	Erected	In Operation	Pounds	Tons	Mesh
Modderfontein B. ^a	80	77	5	5	1,650	14.1	12
New Modderfontein.....	180	143	7	6	1,320	9.8	67
Rose Deep.....	300	261	7	7	1,250	7.9	122
Geldenhuis Deep.....	300	242	7	5	1,133	6.9	231
New Heriot.....	70	54	2	1	1,250	6.7	375
Nourse Mines.....	260	193	7	5	1,268	8.2	183
City Deep.....	194	106	9	5	1,600	11.8	77
City and Suburban.....	160	137	2	2	1,200	6.0	440
Village M. Reef.....	220	168	6	3	1,105	7.0	400
Village Deep.....	180	164	7	5	1,312	8.8	248
Ferreira Deep.....	280	233	7	6	1,137	7.5	258
Robinson.....	250	229	6	6	1,230	7.9	165
Crown Mines A.....	300	246	10	8	1,262	9.6	92
Crown Mines B.....	200	173	6	4	1,212	9.0	106
Crown Mines C.....	160	129	9	8	1,672	15.3	11
Bantjes.....	100	79	3	3	1,415	10.0	62
Durban Roodepoort Deep...	100	91	3	3	1,500	8.6	148

^a The 16 Nissen stamps were not erected until 1914, making a total of 96.

This brings me to the subject of the Nissen or single-unit stamp, 16 of which have just been installed at the Modderfontein B. mill as a result of most favorable trials conducted at the City Deep in 1911.² These trials incontestably demonstrated certain points of superiority of this type of stamp, showing, for example, that pound for pound of dropping weight, the Nissen stamp crushed about 30 per cent. more rock per day and had an increased efficiency of 35 per cent. over the ordinary stamp of equal weight. This stamp has been well described in Mr. Nissen's paper, cited above.

The better results obtained appeared to be due to the elimination of the admittedly weak mechanical features of the older design. The serious defect of the conventional 5-stamp arrangement is that it is impossible to adjust and distribute the feed properly. It is obvious that in the multiple box all the stamps are not doing the same amount of useful work. The complex and mutually opposing currents set up within the mortar box make uniformity of feed impossible. The uncrushed ore is not distributed evenly over the dies, so that coarser pieces are not always most suitably arranged under the falling stamp. This difficulty in controlling feed is undoubtedly largely responsible for broken stems and shoe necks, and the uneven wear on dies. Another serious defect of the multiple principle is that with the long screen and the turmoil of pulp set up within

² *Journal of the Chemical, Metallurgical and Mining Society of South Africa*, vol. x, p. 110 (Sept. 1909).

the box, there is no *positive* discharge of the crushed ore when reduced to the desired size, and only that material adjacent to the screen can be discharged. Another obvious mechanical defect of the 5-stamp principle is that different sections of the mortar casting are subjected to continuous and rapid blows, which produce a rocking tendency eventually loosening foundation bolts. These, if kept tight to insure rigidity, will stretch and eventually break.

The distinct advantages of the single-stamp principle may be summed up briefly as follows:

1. Owing to the mortar being circular, the screen can be extended around the stamp for the greater part of the circumference of the mortar, so that it is equally distant from the stamp throughout its full length. The screen is therefore always at right angles to the direct splash of the pulp, in the most advantageous position.

2. At each blow of the stamp the pulp is forced radially against the screen, so that all particles sufficiently reduced are discharged. Owing to the mortar box being circular, each time the stamp is raised all the material in the mortar flushes to the center, to be struck by the falling stamp. It follows, therefore, that the uncrushed ore is automatically returned to the crushing zone, so that the best conditions of feed are maintained.

3. The blow being always received in the vertical axis of the box, the mortar remains rigid on its foundation. This is noticeable in the case of the Nissen stamps at the City Deep, where, after three years of use, the box is apparently as rigid on its base as it was the day it was fixed in position.

4. One of the important features of the Nissen box is that it can be cast with a minimum likelihood of shrinkage strains, which are so destructive to ordinary 5-stamp mortars. The foundation bolts give no trouble, not being subject to undue strains.

5. Another advantage of the unit principle is the more continuous operation and flexibility of the entire plant, as each stamp can be put out of commission independently.

6. An interesting feature of this type of stamp is the comparatively even and flat wear of the dies, which results from the return wash of the ore to the center of the mortar with each stroke of the stamp, and from the increased number of rotations of the stamp due to wider cam and tappet faces.

The only objection seriously urged against the single-stamp unit in South Africa is the initial cost of erection, as compared with the Californian. On the assumption that the single-stamp installation required a longer mill building than a Californian for a corresponding tonnage, we should require longer bins, greater fall for launders, increased depth of pump pit and higher pump lift. But this objection is practically nullified by experience at Modderfontein B., where tonnages of 14.25 and 29 tons from Californian 1,650-lb. stamps and Nissen 2,000-lb. stamps,

respectively, both using 0.27-in. aperture screen (9-mesh), indicate a negligible difference in length of mill required. It is true that 2,000-lb.

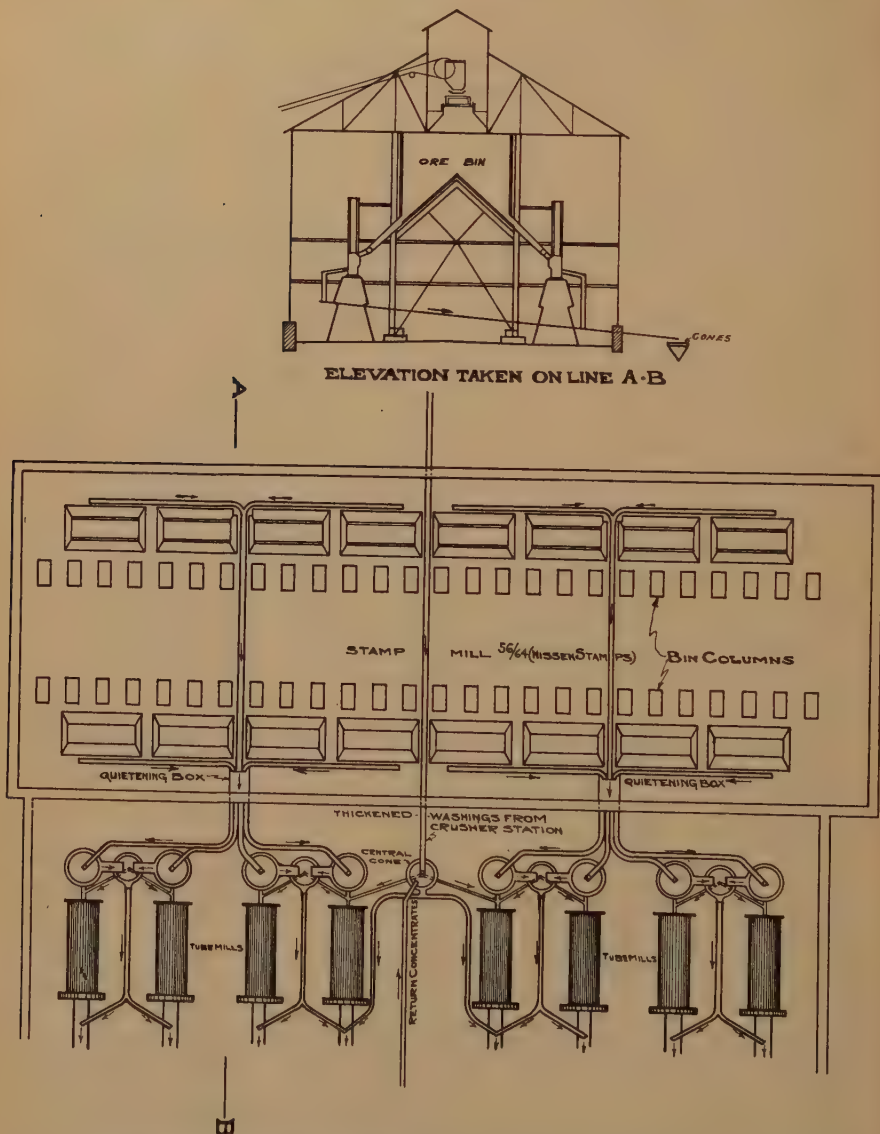


FIG. 2.—FLOW PLAN OF STAMP MILL, PROPOSED NEW REDUCTION WORKS, RAND MINES, LTD.

stamps in multiple arrangement have recorded duties as high as 20 tons; but assuming that 1,650 lb. is the economic mechanical limit for stamps arranged in series of five, the chief objection to the Nissen stamp may be

regarded as unimportant. On the basis of the above data, 50 1,650-lb. Californian stamps would crush 713 tons per day; the same tonnage can be crushed by 24 Nissen stamps. The former would require a mill building 117 ft. long, if arranged in single line; the latter, 133 ft. Even if we compare the room taken up by the two types of stamp of equal weight, the disadvantage of increased length of building for single stamps in this case would be easily offset by improved efficiency.

The floor plan of the proposed new works of the Rand Mines, Ltd., is shown in Fig. 2.

AMALGAMATION

The recovery of gold on amalgamated plates is still a very important department in Rand reduction plants, in spite of its gradual restriction in some cases to a relatively small amalgamating area, and the tentative proposals made from time to time to eliminate it altogether. It would simplify metallurgy enormously, both from a technical and metallurgical point of view, if the cyanide process could be depended upon to extract all the gold from Rand ores. In America, in recent installations, amalgamation is omitted. In these cases, however, the gold is found either (1) so finely divided, though free, that its dissolution by cyanide is sufficiently rapid and complete, or (2) it does not exist in amalgamable form. On the Rand, where we find recoveries by amalgamation as high as 70 per cent., the greater portion of the coarse gold, variable in quantity at different mines, is only economically recoverable on plates; and there is, therefore, no likelihood of this step in treatment ever being dispensed with.

Many factors have influenced and changed amalgamation practice on the Rand. In the early days "inside amalgamation" was carried on in the spacious and specially designed mortar boxes provided with plates; but high mercury losses, the impossibility of obtaining a reliable screen sample, the inconvenience of cleaning up, and the necessity of modifying the design of mortar box to a narrower and more efficient type for crushing purposes, were the causes of the gradual abandonment of this practice. Apron battery plates are still retained in many mills, one to each battery of 5 stamps, and are of the conventional type, the average size being 5 by 12 ft., inclined to a maximum of about $1\frac{1}{2}$ in. to the foot. When tube milling was introduced, making coarser crushing possible, supplementary plates were placed after the tube mills, to offset the reduced recovery from the battery plates, and to recover the gold released in the tube mills. A shaking movement was imparted to these plates, similar to that of a concentrating table, by means of hard-wood springs and an eccentric driven from a countershaft. As many as seven of these plates, 5 by 12 ft., were placed below each tube mill, set at a grade of about 10 per cent. The shaking movement was considered necessary to insure a good distribution and flow of pulp; but as tube mills became more effi-

cient and numerous it was found that a stationary table, set at 18 per cent. grade, was quite as effective; and in many mills the tube-mill plates were changed from shaking to stationary.

The practice at present (Fig. 3) is as follows: Amalgamation with and without battery plates; classification of stamp-mill pulp in tube-mill cones, the underflow of these going to tube mills and thence over tube-mill plates; the overflow going either (1) direct to the cyanide plant, (2) to a separate set of amalgamating plates, or (3) to join the main pulp stream leaving the tube mills. It will be seen that in (1) and (2) only the tube-milled pulp is amalgamated on the secondary plates, while in case (3) all the pulp passes over these plates.

This difference in practice is affected by a number of considerations. We find the Consolidated Goldfields using only three stationary plates to a tube mill, after re-classifying the overflow from the tube-mill cones and bypassing a considerable amount of pulp without secondary amalgamation; while the Rand Mines, in their newer plants, use six tube-mill plates, and bypass little or no pulp. Undoubtedly the former method is the simpler; there is less capital expenditure for plant and buildings, less capital held over in the form of amalgam, and reduced operating expense. But experience on the Rand would seem to show that it is impracticable to fix any definite limit for plate area. By restricting this area to narrow limits an increased burden of extraction is thrown on the cyanide plant, involving an irregular realization from a considerable portion of the total gold extracted, which is undesirable where the maintenance of a uniform and easily available yield is of so much importance from the point of view of mine administration. But apart from this consideration, variations in ore, as regards grade, coarse-gold content, and general composition, certainly affect amalgamation; and as these factors cannot be depended upon to remain constant from month to month even on the same mine, I am of the opinion that the radical reduction in plate area recently advocated can be regarded as safely applicable only in special cases. The amalgamating facilities at Modderfontein B. may be considered a mean between the two extremes; here there are no battery plates, but 30 stationary 5 by 12 ft. tube-mill plates, or a total of 1,800 sq. ft., with capacity of 39 tons of ore per plate per day. Trials have been made at this plant with a view to reducing the number of plates, but the appearance of free gold in the sand residue during these trials, and during periods when the 30 plates were overcrowded by a sudden increase of tonnage, led to the conclusion that, in this instance at least, any reduction of plate area would be inadvisable.

The details of amalgamation do not differ essentially from practice in other parts of the world, which has been described in a number of text books. The plates are periodically "dressed" by brushing up the accumulations of concentrate, or "black sand," with adherent amalgam,

followed by the usual sprinkling with mercury, and vigorous rubbing to produce the necessary uniformity and softness of surface. The operation of "scraping" is carried out once a day, usually in the morning, by means of steel scrapers, which remove the bulk of the gold accumulated during the previous 24 hr. To prevent an excessive accumulation of gold on plates, "steaming" is resorted to at varying intervals, the usual practice being to steam one-third of the plates every month, so that all plates obtain this treatment about four times a year. The procedure is to place a tight wooden cover over the plate and introduce steam until the amalgam is sufficiently softened by the heat. The plate is then scraped until only enough amalgam remains to insure a good surface. All accumulations of amalgam and "black sand" are treated either in a grinding pan or amalgam barrel, the latter being preferred. Grinding with steel balls is carried on for about 2 hr. The contents of the barrel are then run slowly off into a *batea*, where further amalgamation takes place, the residual slimed black sand overflowing to suitable boxes, where it is retained for further treatment. All amalgam from mercury traps, launders, etc., is similarly treated. The resultant product from the grinding barrel and *batea*, when the bulk of the gold has been separated from it in the first operation, is subsequently treated in a small tube mill, whence the outflow is allowed to pass over an amalgamated plate. To recover the major part of the residual gold not saved in these two operations, the slime thus produced is treated either in separate tanks by prolonged leaching, or in a small air agitating vat, followed by decantation. By this means extractions of over 95 per cent. are obtained from this product. The tailing is sometimes discharged into the slime plant and mixed with a current slime charge.

TUBE MILLING

The determination of the exact scope of the tube mill in crushing ore, and the conditions under which it would work most effectively, was not arrived at until some time after its introduction. The application of the diaphragm cone, which made possible a uniform feed of easily controllable moisture, and the introduction of heavy stamps, which permitted coarse crushing within the wide range necessary for fixing, by trial, the economic scope of the tube mill, were important factors in the development of this important auxiliary to crushing, whose value and limitations are now pretty well understood.

The obvious desideratum in distributing the work of crushing was to ascertain the economic point of separation between stamp milling and tube milling in the production of a final product sufficiently fine to yield the maximum net profit. To determine this point of economic balance has been no simple undertaking, but has entailed a vast amount of practical investigation.

The cost of tube milling had an important bearing on this problem. The first step toward reducing operating expense was in the substitution of selected pieces of basket ore for grinding purposes, in place of the imported pebble. Then it became apparent that a tube mill worked more efficiently on coarse than on fine pulp; and the opinions of metallurgists finally converged to the now generally accepted view that the product of a 9-mesh battery screen (0.272 in. aperture) is about the economic limit of size for tube-mill feed on the Rand, in producing a final product most suitable for cyaniding. Beyond this point, except in special cases, the tube mill encroaches on the domain of the stamp.

This point having been satisfactorily settled, it remained to determine the proper ratio of tube mills to stamps. At present this ratio is extremely variable, dependent upon many factors. But in new mills, the usual allowance is based upon the result of extensive trials in which all the leading groups have participated. In recent practice, the tendency has been to increase the ratio of tubes to one 22 ft. 6 in. by 5 ft. 6 in. tube mill to ten 2,000-lb. stamps, or one tube mill to 200 to 250 tons per day of 9-mesh product. At Modderfontein B. the ratio, when the sixth tube mill shall have been erected, will be one of the latter to 264 tons per day of 9-mesh battery product. At the proposed new mill the ratio will be one tube mill to seven Nissen stamps, or one mill to 203 tons of 9-mesh product per day. Latter-day practice aims at the production of certain screen grades in the various stages of reduction, which will give the most suitable final products for sand and slime treatment. The following gradings are fairly representative of the work being done at a modern plant using efficient classification and vacuum filtration:

	+ 60 (0.01 in.)	+ 90	- 90 (0.006 in.)	- 200 (0.0025 in.)
	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Entering tube mill:				
Main circuit.....	85.81	8.08	6.11
Concentrate return.....	58.30	30.87	10.83
Leaving tube mill:				
Main circuit.....	18.74	23.58	57.68
Concentrate return.....	10.59	31.52	57.89
Final pulp before slime separation.....	1.40	13.87	84.73
Sand (39 per cent. of total ore).....	9.28	38.76	40.94	11.02
Slime (61 per cent. of total ore).....	10.00	90.00

(Over the period represented by above gradings, the extraction by cyanide was 93.7 per cent.)

We now come to a consideration of the more important improvements in the details of tube milling, as locally evolved. In reviewing the

progress of this branch of reduction since the general adoption of tube mills by the gold mines of the Rand in 1904 to 1905, it must be confessed that divergence from the practice and design followed in the initial stages has been significant. Up to a few years ago the 22 ft. by 5 ft. 6 in. tube mill, standardized on these fields, remained almost unchallenged; and in the opinion of many, the change since introduced to greater diameter and less length has still to be justified by practical comparative tests. Modifications in methods of introducing the pulp, quantity of feed, percentage of moisture, crushing load, speed of rotation, and in methods of lining have naturally resulted as experience matured, and have all been subjects of considerable controversy here and elsewhere.

The principle of peripheral discharge, abandoned at the inception of local tube milling by reason of excessive wear on liners, went uninvestigated for many years, although the probability of its return to favor was predicted by prominent metallurgists at the time. That there was justification for this assumption has been shown recently, for as the result of trials carried out on a working scale and only lately completed at the City Deep, Ltd., a gain in crushing efficiency has been completely proved when using scoop elevators, fitted at the discharge end of the mill and passing the pulp out through the trunnion in the ordinary way.

A great deal of theoretical work has been done by various investigators in attempting to arrive at a method of comparing the crushing efficiency, or work done, by stamps and tube mills. So far, however, the production of sand of -90 grade remains the only practical standard of comparison which has been generally adopted. As the result of exchange of ideas and experience, the points of difference in local practice have gradually been brought into line, until to-day the procedure—speaking generally—may be considered uniform for standard tube mills using the ordinary central discharge.

In feed devices, the Schmitt spiral lift is now universally adopted at the more recent plants. This appliance is particularly suitable for taking the usual free underflow from the thickening cone, and can handle without difficulty the large tonnage of broken quartz necessary to maintain the load. In tonnage of dry solids fed and percentage of moisture, practice seems to have settled down to a range of 250 to 400 tons per 24 hr., depending upon the coarseness of pulp fed, with a moisture of from 32 to 40 per cent., the latter factor varying directly as the tonnage fed, within the above limits. Local considerations, however, such as physical difference in the ore itself, the occurrence and accessibility of the gold, make it necessary to modify the procedure slightly between mine and mine. Concerning speed of rotation, as the result of tests carried out locally, the tendency has been to reduce the speed to 28 r. p. m. from

the 32 to 33 considered the desideratum a few years ago, giving an average peripheral speed, using an Osborne liner, of about 400 ft. per minute.

In regard to liners, although it has been universally recognized that these have a supremely important bearing on the crushing efficiency obtainable in tube milling, the opinions of the highest authorities the world over have been at variance as to both material and design. As far as the former is concerned, owing to the extremely abrasive nature of the ore dealt with on the Rand, practice elsewhere has had little to do with shaping the final opinion now held here, as to what is considered most suitable. Beginning with shaped silex blocks, imported with the mills, shortly afterward replaced with local chert, through various stages of composite liners, composed of cement and iron, we find the majority of tube mills on the Rand today using the Osborne bar liner, the standard design consisting of tapered steel bars of 4 by $1\frac{1}{4}$ in. to $\frac{3}{4}$ in. section, set longitudinally, the thick edge being held in position against the shell of the mill by wedging with flat bar iron $2\frac{1}{2}$ to 3 in. wide by $\frac{1}{2}$ to $\frac{3}{4}$ in. thick. This has proved superior to all other liners in efficiency and longevity, the extra work accomplished being greatly in excess of the increase in power consumed. Moreover, the average variation in the internal diameter of the liner during its life is much less than with either the silex or the composite-block type, which start with a thickness of 6 in. or over.

Undoubtedly the most interesting and practical development of latter-day tube milling is to be found in the results of experiments previously referred to, which had, as their original object, the investigation of the principle of peripheral discharge. The effect of obtaining such a discharge by means of a scoop or elevator is to change the nearly horizontal line of pulp level, determined by the diameters of inlet and outlet, to a sloping line from the inlet to the lowest point in the circumference of the opposite end. Thus the mill, with the same feed and pebble load, is working on a considerably smaller pulp load, with the result that a comparatively greater grinding surface is effectively exposed, resulting, as might be expected, in an increase of production of —90 product as well as of horsepower consumed. In carrying out these trials a standard 22 ft. by 5 ft. 6 in. mill was employed, fitted with a scoop discharge, in which the radius of the lift circle could be varied. These trials were conducted in two stages: In the first, the scoop was given a maximum lift, in this case about 28 in. radius. In this series, the power consumption, wastage of pebble load, influence of feed, crushing and mechanical efficiency, were investigated. (C. E. = production of —90 in tons per mill. M. E. = $\frac{-90}{\text{hp.}}$.) In the second series, the radius of the lift circle was gradually reduced, and a comparison made with the previous results. In

both trials the screening used in the stamp mill was 64 to 100 meshes to the square inch. The conclusions deducible were as follows:

1. That when maintaining the pebble load at the center mark with feeds ranging from 250 to 400 tons per 24 hr., a 25 per cent. increase in crushing efficiency could be obtained, but with proportionately increased power consumption.

2. That within certain limits of feed, the weight of the pebble load can be decreased by 25 per cent. without affecting the crushing efficiency, with about a 10 per cent. decrease in power consumption, the mechanical efficiency showing a corresponding increase.

3. That the wear on the crushing load is increased 300 to 400 per cent.

4. That by decreasing the effective radius of the scoop from the maximum possible (about 59 in. in a 5 ft. 6 in. mill) to the ordinary trunnion discharge, a steady decrease in crushing efficiency is accompanied by a proportionate decrease in power consumption.

Considerable additional wear of liner would naturally result when running under these conditions; the ratio of the increase, however, was not determinable during the trials. Apart from the obvious fact that there is a considerable saving in head, it remains to be proved whether the lowering of the pulp level in the tube mill is best done by using an elevating scoop or by passing the pulp through openings in the periphery, as originally practiced by Davidson. Locally, the former system has an overwhelming advantage, owing to the serious alteration to existing plants that would be necessitated by the latter. With the use of the pulp elevator, or scoop, the enormously increased pebble feed would have to be faced, amounting to 25 tons per mill per day. This would necessitate proper additional provision being made for sorting, conveying, and feeding, which would mean a practically insurmountable difficulty in existing plants. On the other hand, the effective radius of the elevator can be reduced at will, any increase in crushing efficiency between normal and 25 per cent. being obtainable.

An interesting innovation in the design of recent plants, first introduced at the East Rand Proprietary, is the arrangement shown in Sketches C and E, Fig. 3, wherein the tube mills are placed below ground level, thus permitting flow of pulp by gravity from stamps to tube-mill cones, and saving the excessive wear on pumps. This is desirable even though it entails a subsequent higher lift of finer pulp, more suitable for pump elevation.

The latest design of tube mill used by the Rand Mines (Fig. 4) includes a ball chamber interposed between the main crushing section of the mill and the outflow trunnion. This chamber is provided with a cast-iron step lining bolted to the shell, and the grinding medium used is 10-lb. steel balls, of which about 20 are required. The object of these balls is to reduce the small spent pebbles which are being continually

rejected by the mill. The usual practice is to remove these from the pulp by means of a trommel fixed to the discharge trunnion, and periodically convey them to the stamp battery. The ball chamber is very effective in doing away with this nuisance, and has not been found to reduce the grinding efficiency of the mill, owing evidently to the additional work done by the balls, nor do the balls wear out unduly.

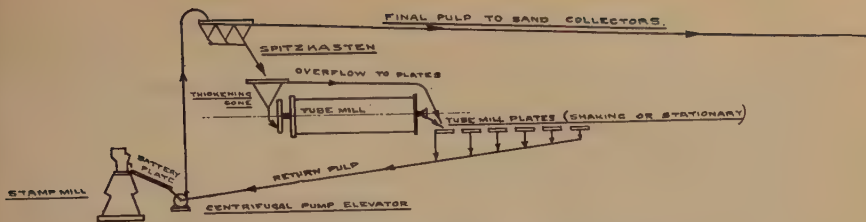
CLASSIFICATION

The necessity of providing a specially thickened and classified pulp for tube mills gave fresh impetus to the study of classification, which hitherto had been chiefly confined to the rather crude methods in vogue of separating sand from slime. The earliest classifiers were of the inverted pyramid type, a first series of spitzlütten with small pockets being designed to eliminate the coarse sand and concentrate, which were collected and given a special treatment, and a second series of much larger pockets being used for separating sand from slime. A further separation of sand and slime took place in the sand collectors, the overflow of which passed to a series of return-sand spitzkästen for the further elimination of sand, the final overflow product from the latter going to the slime plant.

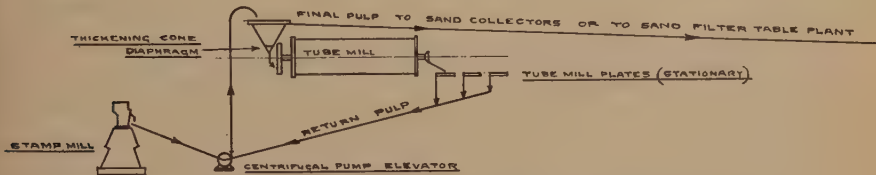
When tube mills were introduced in 1904, the coarse product was no longer separately collected but run to these mills; and the apparatus for classification gradually took the form of a simple series of spitzkästen, provided with underflow nozzles of different apertures. This underflow passed to the tube mills, while the overflow went direct to the sand collectors, in which the slime was separated from the sand and discharged by means of adjustable overflow weirs into so-called return-sand classifiers, which in turn discharged their overflow to the slime plant.

The first step in the much-needed improvement in classification was taken in 1908, when Messrs. Caldecott and Smart developed what is known as the diaphragm cone, now generally employed for thickening and classifying tube-mill pulp. This consists of a sheet-steel cone, 5 to 6 ft. in diameter and 7 to 9 ft. deep, provided near the apex with an iron disk or diaphragm. This disk is 8 to 10 in. in diameter, and the annular space between it and the sides of the cone is 2 to $2\frac{1}{2}$ in.

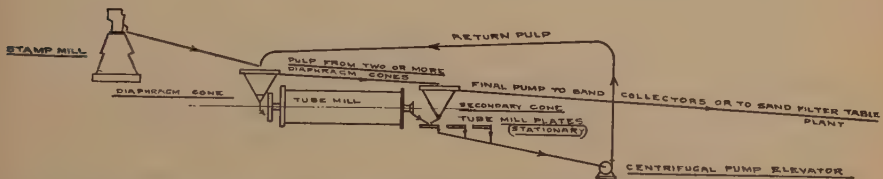
The proper action of this diaphragm is obtained only when the cone is full of solids. The mass of sand in the cone then assumes a concave surface, the deepest point from the plane of the overflow edge being immediately under the central pulp inflow. At this point, where the coarsest and heaviest product accumulates, the surface is seen to be in a state of slow continuous subsidence. This slowly subsiding mass of heavy material, conceivably irregular or conical in shape, may be presumed to act as a descending wedge, retarded in its downward course by the supporting diaphragm; while the finer sand, with the slime, tending



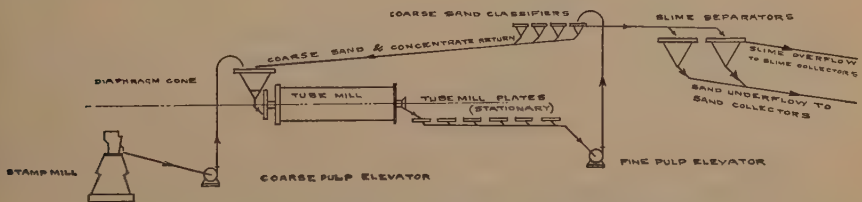
A.—Common arrangement, still retained in many plants.



B.—Arrangement used by Consolidated Goldfields.

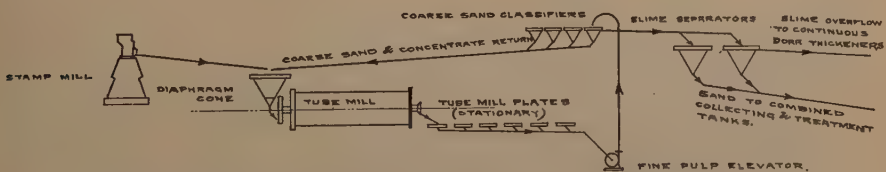


C.—Improved arrangement used by East Rand Proprietary and Consolidated Goldfields, the only difference being that the former use more tube-mill plates and do not use the secondary cone.



D.—Improved arrangement used by Rand Mines, Ltd.

NOTE—At the City & Suburban mill, battery plates are used and the pulp from the coarse sand classifiers goes to a Caldecott sand-filter plant.



E.—Improved proposed arrangement for new mill, Rand Mines, Ltd.

FIG. 3.—DIAGRAM OF VARIATIONS IN CLASSIFICATION ARRANGEMENTS.

to adhere to the sides of the cone, would appear to be literally pushed aside and crowded upward by the central column, so reaching the overflow rim and escaping. The slow, thick stream issuing from the apex carries only from 26 to 30 per cent. moisture, and a minimum of the fines which it is desirable to exclude from the tube mill.

These cones have a large capacity, and, barring the one disadvantage of vertical height required, may be considered the most simple and suitable means yet devised for providing all the conditions requisite for a tube-mill feed that will enable the mill to work at highest efficiency. This device is now used either as the sole means of classifying mill pulp, as in B, Fig. 3, or in conjunction with hydraulic classifying cones, as in D and E, Fig. 3. An important aspect of this innovation was its having made possible the introduction, in 1907, of the Caldecott sand-filter table, a device whose success obviously depended upon securing a suitable thickened pulp, containing a small amount of water and a minimum of slime. The primary object of this appliance, as explained by the inventor,³ was to obtain sand in such a condition for treatment as would warrant the elimination of sand-collecting vats, which could then be used for treatment purposes. Obviously, the effect of this was to increase very considerably the capacity of a leaching plant.

Metallurgically considered, however, the significant feature of this appliance was its revival of the old question of the possibility of single treatment of sand after proper classification. In this connection it is interesting to note a prediction made by Charles Butters in 1895, that "the whole question of double treatment really resolved itself into a matter of filling the vats with clean stuff, and he was confident that the day would come *when there would be no double treatment.*" The first notable success in America in collecting and treating sand in the same vat was at the Homestake mill; but in Africa, with the single exception of the East Rand Proprietary, double treatment has been retained until quite recently.

In 1910, when the new plant for the Modderfontein B. mine was being designed, I undertook to evolve a simpler method than the filter table for obtaining a clean sand, with a view to the subsequent elimination of separate collecting tanks. This classifying plant consists of eight small primary hydraulic cones, 2 ft. 9 in. in diameter and 2 ft. 6 in. deep (see D, Fig. 3), designed as concentrators for insufficiently ground sand particles from the tube mills. The overflow from these gravitates to four larger hydraulic cones, 8 ft. in diameter by 6 ft. 9 in. deep, which effect a very satisfactory separation of sand from slime, the overflow gravitating direct to the slime collectors. The underflow of the large cones is evenly distributed in the collectors by the Butters and Mein

³ *Journal of the Chemical, Metallurgical and Mining Society of South Africa*, vol. x, No. 2, p. 43 (Aug., 1909).

distributor, a device recently revived on the Rand Mines group after several years of disuse.

This system, in view of the possibility of treating a considerable amount of fine (-200) sand in the Butters filter, was found to be well

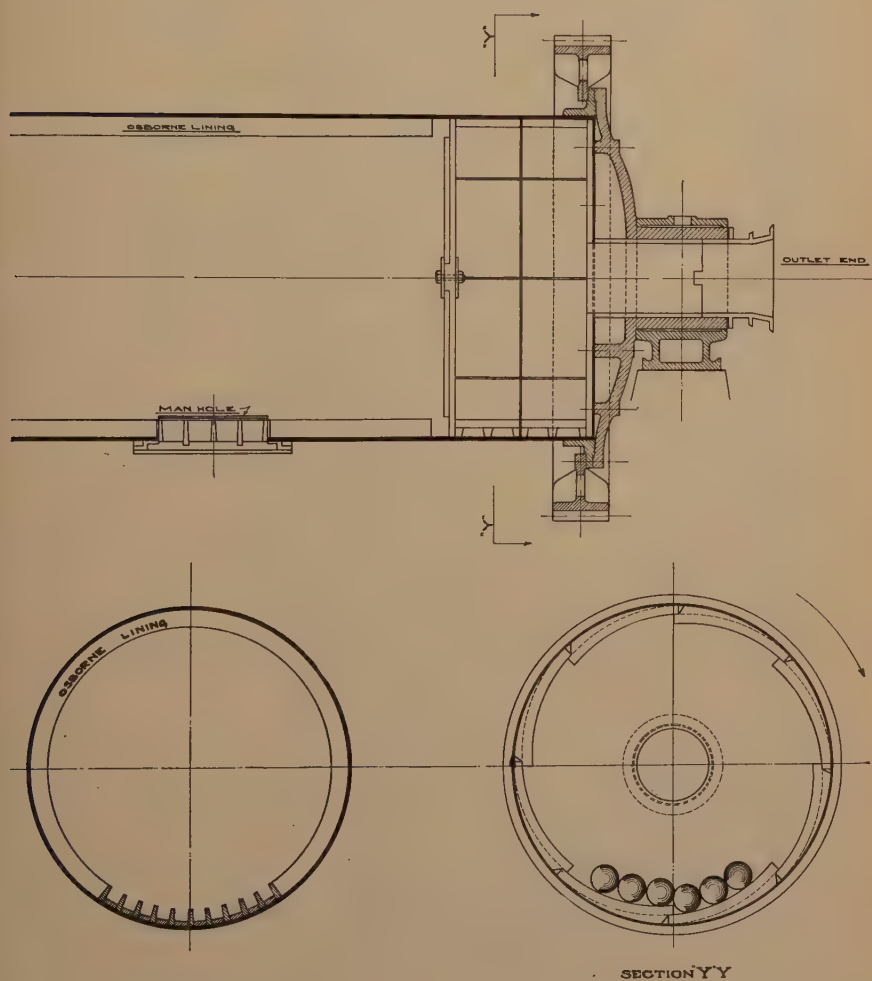


FIG. 4.—TUBE MILL WITH BALL CHAMBER, MODDERFONTEIN B. REDUCTION WORKS, RAND MINES, LTD.

adapted to this mode of filtration, the correct proportion of fine sand and slime being easily obtainable. Moreover, an evenly distributed sand charge was secured, free from lumps and layers of slime. This system has been adopted by other mines of the Rand Mines group; and in newer plants has, with a few modifications, superseded the primitive method of

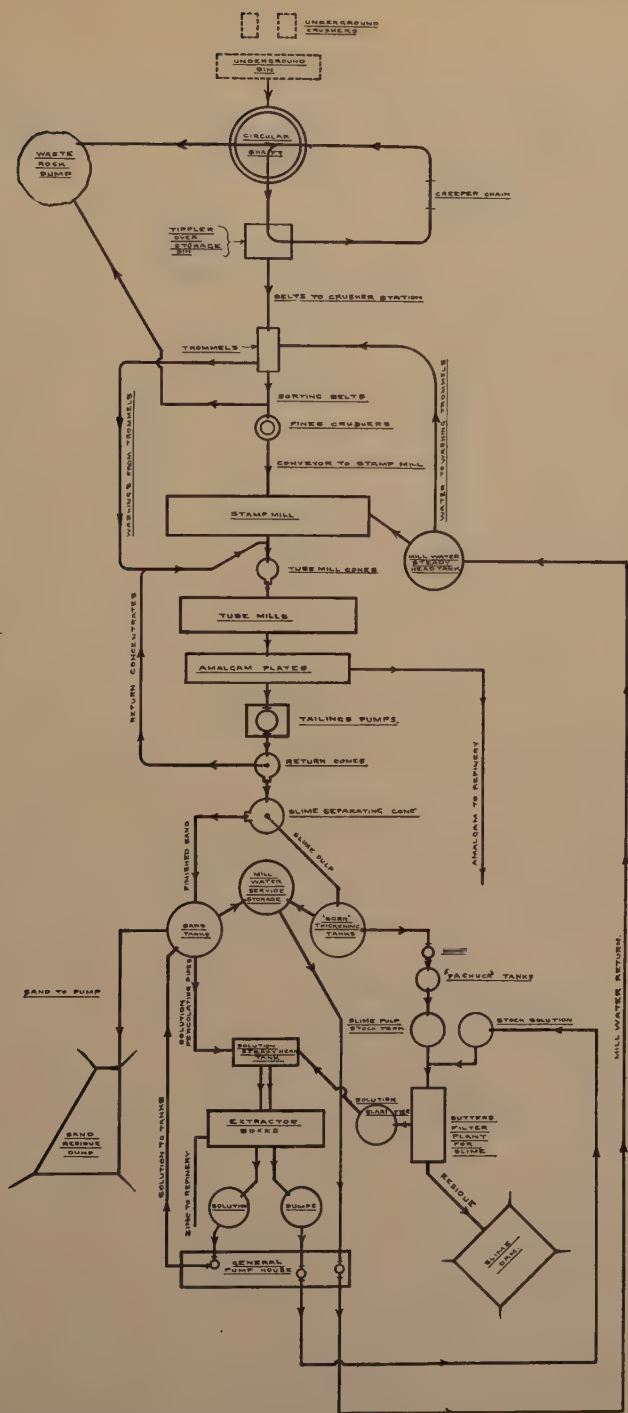


FIG. 7.—FLOW SHEET, PROPOSED NEW REDUCTION WORKS, RAND MINES, LTD.

charging sand and slime together into collectors and depending upon a Kaffir and a movable hose for even distribution.

At the Crown Mines, an improved hydraulic attachment for cone classifiers was devised by H. Brazier, the reduction foreman at "C" mill. This consists of an adjustable nozzle for discharging water in proximity to the apex of the classifying cone in the form of a thin circular sheet, directed horizontally between the nozzle and the cone. By means of this system of cone separators, the classification of sand is permissible within a wide range, and has enabled us to obtain a charge of sand so permeable and uniform that after practical trials the separate sand collector is now recognized to be superfluous, and will be omitted in forthcoming plants to be erected by the Rand Mines group (Figs. 5, 6, and 7).

We have then at the present time on the Rand two very satisfactory methods for classifying sand: (1) the Caldecott sand-filter table, and (2) the system of hydraulic cones just described. The former possesses the advantage of delivering a solution-borne sand to the treatment tank, and so shortening the time of treatment and increasing the capacity of existing plants, which is a very desirable thing, especially where the saving of capital outlay is imperative, as at the City & Suburban, where the sand-filter table obviated an expensive and awkward extension of plant. But for new mills adopting single treatment of sand, the indirect saving in sand plant would appear to be offset by the lower initial expense of the hydraulic-cone system itself, the small amount of attention required, and its negligible cost of maintenance and operation, as compared with the filter table.

In E, Fig. 3, an improved arrangement of the two series of cones is shown. Differing from general practice, the slime separators are placed near the ground, only just high enough to permit their overflow to run to the Dorr thickeners, while the underflow of sand is elevated to the sand-treatment tanks. We thus have but two lifts in this type of plant: the first, and highest, elevating the tube-milled pulp to the primary classifiers; and a much shorter lift elevating sand and water only. This may be seen in Fig. 7, where, however, the treatment tanks are diagrammatically shown at a considerable elevation, whereas they will stand only just high enough above the ground to allow room for the residue trucks.

TREATMENT BY CYANIDE

It was only after the introduction of the cyanide process that the distinctive terms "sand" and "slime" came into common use. In the early days of cyaniding the bugbear was slime, or the unleachable, finely divided portion of the mill pulp, which the mill man endeavored to avoid making, in order to secure as large a proportion as possible of leachable sand. Until a method of treating slime was devised, this

product was impounded in dams; these accumulations have now mostly been treated.

When the decantation process made it possible to deal with current slime direct from the stamp mill, the production of the latter was no longer avoided. Until vacuum filtration was introduced, the slime represented between 30 and 40 per cent. of the total product of crushing; but this proportion has gradually increased with the progressive improvements in treatment, until now as high as 60 per cent. of the pulp is treated as slime. We may say, therefore, that tube mills, improved classification, and the vacuum filter have made possible (1) the treatment of a greater proportion of slime than formerly, which has improved the total extraction, and reduced the cost of treatment, since slime can be more cheaply treated than sand, and (2) the treatment of more finely crushed sand, with consequent improvement in extraction. In the best plants, a recovery of 90 per cent. of the gold from sand and 93 per cent. from slime is now being obtained, or a total recovery of 96 per cent., which is probably the highest extraction economically attainable on the Rand. The metallurgist in Johannesburg to-day is therefore chiefly concerned with those improvements in appliances, general technique, and administration, which, in view of the much reduced grade of ore, will minimize cost of treatment.

TREATMENT OF SAND

The treatment of the sand by leaching with cyanide solution in steel tanks, ranging in size between 45 and 56 ft. in diameter, is still the accepted practice. In the early days an attempt was made to collect and treat sand in the same tank, but an imperfect knowledge of classification, or a failure to recognize its importance, made this scheme impracticable. Until very recently, the prevailing practice was to collect the sand in a series of tanks, known as "collectors," from which it was shoveled out by natives and transferred to the treatment tanks. The majority of plants were built with the collector superimposed on the treatment tank, but this arrangement was finally superseded by the erection of collectors and treatment tanks on the same level, the transference of sand from one to the other being by means, first, of trucks, and later, by belt conveyor. The latter was supposed to be cheaper in first cost, as it eliminated the expensive steel superstructure for supporting the enormous weight of superimposed tanks, and also reduced the height of pulp elevation. But it would appear now, after a pretty thorough experience with both systems, that, as regards first cost, the superimposed system has a slight advantage; while owing to the rapid disintegration of conveyor belts in the dry atmosphere of the Rand, and the considerable maintenance cost of conveyor systems in general, the superimposed

tanks are less expensive to operate, in spite of the higher pulp lift required.

At the present time there is still considerable variation in the modes of collecting sand. The following methods are in use:

1. In the older plants all the mill pulp is run into a collector, through a rubber hose 4 to 6 in. in diameter, manipulated by a Kaffir who moves about in the tank changing the position of discharge to prevent undue slime accumulations; the slime and water, with varying proportions of fine sand, overflow through discharge gates provided with an adjustable canvas blind, which is raised to suit the overflow as the tank fills.

2. The total mill pulp is distributed to the collector by means of a peripheral launder provided with outlets; the water and slime overflow through an adjustable opening at the center.

3. The sand is classified and thickened in diaphragm cones, dewatered on a sand-filter table, and delivered in cyanide solution either to a collecting or treatment tank by means of a Butters and Mein distributor.

4. The sand is classified in hydraulic cones and delivered in water to a collecting tank by means of a Butters and Mein distributor; in the plant shown in Fig. 7, the sand will go direct to the treatment tank.

It was formerly customary in some plants to give the sand a preliminary treatment with a weak cyanide solution in the collectors; this practice has now been generally abandoned, and the only operation that takes place in the collector is the forced drainage of the charge by means of pumps, thus reducing the moisture to about 14 per cent. The sand is discharged from the collectors by (1) hand shoveling through discharge doors into treatment tanks or to belt conveyors; (2) by means of the Blaisdell excavator. The high capital outlay required for the latter, without compensating economy in operation, has led to the retention in newer plants of the older system of hand shoveling by natives. All plants using a belt conveyor from collectors to treatment tanks have, however, retained the excellent Blaisdell distributing mechanism for distributing the sand in the treatment tank.

The system of applying cyanide solution to the sand does not differ essentially from practice elsewhere, and need not be particularized here. The standard strength of strong solution used ranges between 0.10 and 0.25 per cent., depending upon local conditions, the tendency in recent years being to use weaker solutions than formerly. From 6 to 8 days' contact is usually allowed in the treatment tanks, and about 2 parts of solution to 1 of ore is the average quantity required for leaching purposes.

The methods in vogue of disposing of the residue are by hand shoveling or excavation with the Blaisdell machine into (1) trucks, (2) Bleichert aerial conveyor, or (3) belt conveyor. The second method is used at

only one property, the Brakpan mines, where it is reported to be giving satisfaction; the third method is used on two mines, but whatever advantages it may possess have not been generally recognized; while the first method, namely, truck haulage by surface cable, is in general use. It is considered the cheapest, most flexible, and best adapted to local conditions, and will probably continue to be the favorite means of disposing of sand residue.

TREATMENT OF SLIME

Since 1894, the continuous treatment of slime by the so-called decantation method has meant the recovery of over 80 per cent. of the gold from a very considerable portion of the total ore crushed, and has been retained by the majority of operating mines. In most cases, however, it has been retained, not because its limitations have not been recognized, but because possible improvements in recovery do not always justify the sacrifice of capital involved in replacing old plants with new, especially on mines of short remaining life.

The technique of decantation, which is well described in several technical works, need only be briefly touched upon here. The slime pulp from the classifiers is settled in what are known as collectors, large cone-bottom tanks, provided with an overflow rim and adjustable decanter, in which the slime settles to the bottom, the water escaping at the overflow rim. The settled slime charge after decantation, containing approximately 50 per cent. of moisture, is then sluiced out with a weak cyanide solution into the intake of a centrifugal pump connected with the center of the cone bottom, and transferred to another tank, known as the "first settlement tank." Here the charge is kept in circulation by means of a pump, then allowed to settle, as in the collector, and the gold-bearing solution is decanted off to the zinc boxes. This decanted charge is again transferred, subjected to a similar settling and decantation in a "second settlement vat," and finally discharged to the dam with water or a very dilute precipitated solution. Settlement in these tanks is hastened by the use of lime, which is applied in a variety of ways, being either periodically fed to the stamps or tube mills, or to the slime pulp in slaked form, by means of feed hoppers, or as milk of lime from a grinding pan.

The limitations of decantation are obvious: The displacement of the dissolved gold is only practicable within certain economic limits, so that a certain definite gold loss in the final discharge to the dam must always be reckoned on. This loss is necessarily variable, depending upon capacity, difficulties in settlement, and general design of plant. On the other hand, the process is simple and comparatively inexpensive in operation, and, in spite of the losses through imperfect washing, is pecu-

liarily applicable to the low-grade slime of the Rand. Even the advocates of more exact and positive methods of treatment must admit that in the latest and best designed decantation plants, in which large capacity is allowed for settlement and dilution, and all other conditions are favorable, the margin between results so obtained and the net results of filtration is not a wide one.

The introduction of the so-called Usher process was an attempt to improve upon decantation by removing the gold-bearing solution by means of continuous upward displacement. The treatment tanks were provided near the bottom with a large number of perforated pipes radiating from a central upright pipe. The precipitated solution was introduced into this central main from a steady-head tank and allowed to issue slowly through the holes in the radial pipes into the bottom of a partly settled charge. The speed of flow was just sufficient to keep the slime in suspension and permit of a clean overflow at the tank rim. The chief advantage claimed for this system was the reduction in plant required by reason of the time saved in avoiding decantation and settlement. But certain insuperable mechanical difficulties developed in connection with this process which offset its alleged advantages, and it was subsequently replaced by either decantation or filtration.

The first important innovation in slime treatment was the introduction of the Butters vacuum filter in 1910. The essential features of this device are too well known to require description here. There was from the first no question as to the additional recovery obtainable by the filter; the doubtful point was whether the cost of operation would offset the gain in recovery to the extent of nullifying its advantages. This, however, has not proved to be the case. The prevailing cost of operating the filter on the Rand is from $2\frac{1}{2}$ d. (5c.) to 4d. (8c.) per ton of slime filtered, depending upon tonnage treated; and the additional gold recovered ranges from 6d. (12c.) to 2s. (48c.) per ton, depending upon the efficiency of the decantation plants superseded by the filter.

The vacuum filter is now generally admitted to be applicable in the following cases: (1) Where it is desirable to extend a decantation plant of inadequate capacity or obsolete design; (2) where difficulties in settlement make it impracticable to use decantation without prohibitive extension of plant; (3) in all new plants. The most notable instances of (2) were the case of the Randfontein Central mill, where a comparatively new decantation plant treating 2,000 tons of slime per day was replaced by a Butters filter, resulting in a considerably increased recovery of dissolved gold; and the case of the Robinson Gold Mining Co., where the expenditure of £32,000 in 1911, on filter and Pachuca agitators, resulted in an increased net profit of £2,500 per month.

The Butters filter has been adopted in all new plants erected within the last five years. The following Rand mines have adopted the filter:

Butters Filter Plants in the Witwatersrand District

Plant	Number of Filter Leaves	Capacity, Tons per Day
Crown Reef.....	310	900
Brakpan.....	348	1,000
New Modderfontein.....	348	1,000
Bantjes.....	240	600
Randfontein South.....	89	250
Robinson.....	468	1,350
Modderfontein B.....	135	550
Langlaagte Estate.....	117	350
Crown Reef Extension.....	125	400
Randfontein Central.....	532	2,000
Van Ryn Deep.....	310	1,000
Geduld.....	190	500
Government Areas.....	412	1,300
Modderfontein Deep.....	250	700

The slime treatment in these modern plants consists in settling the slime in the same type of settlers as is used in decantation, namely, in tanks as large as 56 ft. in diameter, provided with cone bottoms, decanters, and overflow launders. When the charge is settled and decanted down to about 50 per cent. moisture, it is transferred with cyanide solution to the Pachuca tanks. These tanks are of standard size, 15 ft. in diameter and 45 ft. deep, and hold between 80 and 100 tons of slime (dry weight), depending upon dilution. This agitator, requiring air representing in volume and pressure an expenditure of about 3 hp., is considered, by reason of its simplicity and low operating and maintenance cost, to possess advantages over the various mechanical types, in spite of its great height. Agitation is continued from 6 to 8 hr., and the pulp is then transferred to a large storage reservoir, whence it is delivered to the Butters filter plant, as required.

Two modifications of this procedure will be adopted in the next plant to be erected by the Rand Mines, Ltd. The old system of intermittent slime settlement will be replaced by continuous settlement in Dorr thickeners. In trials conducted with this device it was found impossible to reduce the slime to the same low moisture as the intermittent system, without a much larger capacity than is allowed in American mills, where this device is chiefly applied to the much more simple settlement of slimed mill pulp containing a considerable quantity of fine sand. To compete with intermittent slime settlement on the Rand, a Dorr thickener 40 ft. in diameter cannot handle more than 150 tons of slime per day; but even so, in view of the labor and power consumption involved in handling intermittently settled charges, and the high capital cost of slime settlers, the continuous system would appear to possess distinct advantages.

The continuous agitation of slime is made possible by the system just described; and is particularly well adapted to the newer plants where the

height of the Brown agitators can be utilized for the required gravity flow into the Butters stock tank. This system of allowing pulp to flow slowly through a series of agitators was first applied in Mexico; its first application on the Rand was at the East Rand Proprietary about two years ago. It seems likely that all plants to be erected in the future will adopt continuous slime settlement and agitation, followed by vacuum filtration.

PRECIPITATION

The use of filiform zinc for precipitating gold was introduced on the Rand by MacArthur in 1890. At first no difficulty was experienced in the deposition of gold from the stronger cyanide solutions required for sand treatment; but when it was found possible to recover gold from slime with much weaker solutions, precipitation on zinc became more difficult. It was at this time that the Siemens-Halske electrolytic process, which was more effective than zinc in dealing with these weak solutions, threatened to replace the older method; but owing to serious defects in operation, already touched upon, it was finally abandoned in favor of the zinc method, which had been rendered much more efficient by the immersion of the shavings in a solution of acetate of lead, preparatory to filling the extractor boxes. With this exception, precipitation practice on the Rand does not differ materially from that in other mining districts where filiform zinc is used.

The zinc shavings are cut in the usual manner to a thickness of about $\frac{1}{500}$ in. and 1 cu. ft. of such filaments loosely packed, per ton of solution per 24 hr., is the average allowance for capacity of extractor boxes. These boxes, usually built of steel, are from 4 to 6 ft. wide, and of corresponding depth, with 6 to 10 compartments. The existing system of gold precipitation, though highly efficient on the dilute solutions used on the Rand, admittedly possesses many awkward features which have for years stimulated investigation with a view to devising a more compact, positive, and less wasteful substitute. Its weak points are (1) the great area required for plant; (2) the labor required in dressing and cleaning up extractor boxes; (3) the uncertainty of cleanup, owing to the variable distribution of gold not immediately recoverable; (4) the impossibility of recovering at once all gold deposited within a given period; (5) the enormous loss of zinc in the destructive process of recovering its gold content.

The Merrill zinc-dust method, as perfected in America, offered certain distinct advantages over the older process. It is neater and more exact in operation, requires less labor, and possesses the very attractive feature of yielding a complete cleanup of all the gold deposited. After practical trials this process was introduced at three Rand mines: the Brakpan, New Modderfontein, and Modderfontein B.

At New Modderfontein the conditions of precipitation are somewhat peculiar, and imposed a very severe test on the Merrill process. At this plant a solution of under 0.01 per cent. in KCN is effective in dissolving gold in the slime. The resultant gold-bearing weak or slime solution is comparatively simple in composition with a very low metallic and cyanide content, which rendered it extremely difficult to precipitate satisfactorily on zinc dust without undue additions of cyanide and lead acetate. Various measures were tried to promote galvanic action and enliven this inert solution, but without success; and at this plant it was finally decided to revert to the older zinc-shaving method. The process has, however, been retained at the other two properties mentioned.

At Modderfontein B., only one of the two 40-frame, 52-in. Merrill presses installed is required to precipitate 1,900 tons of solution per day—an unusually high duty. The average gold content of the weaker solution treated is 2 dwt., and its cyanide strength 0.018 per cent. The consumption of zinc for the last six months at this mine (June to November, 1914, inclusive) was 0.180 lb. per ton of ore milled, or 0.145 lb. per ton of solution precipitated.

At Brakpan, three 40-frame Merrill presses are in use, precipitating about 2,800 tons of solution per day. Here the consumption of zinc during the last three months (September to November, 1914) is given as 0.152 lb. per ton of solution precipitated and 0.251 lb. per ton of ore milled. The following details covering these three months' operations have been kindly submitted by the Brakpan Mines, Ltd.:

	Tons Precipitated (Average per Day)	Gold Content of Inflow	Gold Content of Outflow	KCN Content Inflow	KCN Content Outflow	Alkalinity (CaO) Inflow	Alkalinity (CaO) Outflow
		Dwt.	Dwt.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Sand solution.....	269	3.952	0.045	0.070	0.036	0.0230	0.027
Medium sand solution.	769	1.548	0.056	0.017	0.009	0.0146	0.015
Slime solution.....	1,791	1.271	0.048	0.010	0.007	0.0149	0.015

The consumption of zinc in the Merrill presses is approximately $\frac{1}{10}$ lb. less per ton milled than in extractor boxes. The cost at Johannesburg of cut shavings is 4.2d. per pound, of zinc dust 3.93d. per pound, so that as regards zinc consumption the presses have the advantage. But this is offset by the cost of additional cyanide required to strengthen the slime solutions sufficiently for good zinc-dust precipitation, which was found to be, at New Modderfontein, 1d. per ton milled.

The general opinion in regard to zinc dust, after three years' experience with the process, is that in effecting economies in zinc consumption and labor, and in affording a complete cleanup of gold, it has fulfilled the claims made for it. On the other hand, it requires more vigilance and

care in manipulation than the zinc-shaving method, is liable to erratic fluctuations in efficiency without assignable cause, and requires the use of stronger solutions than are actually needed for dissolving purposes.

One cannot escape the conclusion that zinc in any form is far from being the ideal precipitant for gold. In 1913, about 9,000,000 lb. of zinc were consumed by the mines of the Rand. When we consider that the greater part of this irrecoverable loss is due to the destructive method employed in separating the gold from the zinc after deposition, it is evident that the existing system is an extremely wasteful one. For this reason, the subject of gold deposition presents one of the most profitable fields for investigation in the whole realm of metallurgy, and I venture to predict that in this stage in the reduction of gold ores, the most important advances in the future will be made.

CLEANING UP AND SMELTING

Local methods of cleaning up the zinc-gold precipitate do not differ essentially from practice elsewhere. The difference in practice between local mining groups is one of minor details only, depending mainly on design and capacity of plant. Briefly, the procedure consists in periodic removal of zinc-gold "slime" and broken zinc or "shorts," and the renewal of the displaced zinc by fresh shavings. The amount of zinc removed from the boxes at each cleanup depends upon the balance desirable between monthly realization of gold and zinc consumption. A minimum of gold reserve in extractor boxes is attainable only at the expense of zinc consumption; hence the recorded fluctuations in the latter between 0.30 lb. and 0.50 lb. per ton of ore treated.

The usual method of destroying the zinc in lead-lined tanks with sulphuric acid is in use on the Rand. A cheap source of sulphuric acid now in common use is the bisulphate of soda (NaHSO_4), a byproduct from the local manufacture of sulphuric acid used in making nitroglycerin. It contains about 30 per cent. of available sulphuric acid. Filter presses are in common use for removing the zinc sulphate from the gold slimes, and for the subsequent washing with hot water, preparatory to drying and smelting.

The two methods of smelting followed on the Rand are (1) direct crucible smelting and (2) lead smelting. The former possesses the important advantage of eliminating the gold absorption which takes place in the furnace bottom of the Taverner or lead furnace, and which is not recoverable until it is necessary to rebuild the furnace; on the other hand, the lead furnace costs somewhat less to operate. Both systems have their good points, and both will probably continue to be used as long as zinc is retained as the precipitant for gold.

PERCENTAGE EXTRACTION AND TREATMENT COST

In concluding this paper a few remarks on extraction and cost of treatment on the Rand may be of interest. It is significant of the steady, consistent progress made in metallurgy during the last two decades that the recovery of gold has risen from 60 per cent. or thereabouts (1890) to a maximum of 96 per cent. (1914). The chief contributing factors were: The introduction of slime treatment (1894); the advent of the tube mill (1904); improvements in classification (1907 to 1910); and finally the introduction of the vacuum slime filter (1909). Concurrently, many improvements in detail, both mechanical and metallurgical, have had much to do with the general advance.

Similarly, in regard to the steady reduction in cost of treatment, many influences have been at work, such as improved efficiency achieved in crushing with heavy stamps and tube mills, improvements in design of plant and change in supervision and general organization. The cost of reducing ores on the Rand ranges between 2s. 10d. (68c.) per ton milled and 5s. (\$1.20), depending upon capacity and efficiency of plant. As an instance of low reduction cost I have chosen the case of Modderfontein B., a modern, medium-size plant, showing first the average cost during 1913, when the tonnage was comparatively small, and during November, 1914, when the capacity of the plant had been increased.

Cost of Ore Treatment at Modderfontein B.—Average for 1913

Average Tons per Month Milled = 33,715

	Per Ton Milled Pence	Cents
Ore sorting and breaking:		
Including white wages, colored wages, colored labor sundries, machinery and spares, sundry stores, workshops, power, and sundries	3.429	6.8
Transport from crusher station to mill bins:		
Including white wages, colored wages, colored labor sundries, stores, workshops, power, and sundries	0.918	1.8
Stamp milling:		
Including white wages, colored wages, colored labor sundries, mill spares, shoes and dies, water, sundry stores, workshops, power, and sundries	8.118	16.2
Tube milling:		
Including white wages, colored wages, colored labor sundries, liners, tube-mill spares, water, sundry stores, workshops, power, and sundries	7.890	15.8
Amalgamation:		
Including white wages, mercury, sundry stores, workshops, assaying, retorting, and smelting	2.163	4.3
Cyaniding:		
Including white wages, colored wages, colored labor sundries, water, cyanide, lime, zinc, chemicals, trucks, rails and fittings, conveyor belts, sundry stores, tailing contractor, workshops, power, assaying and smelting, stables, and sundries	19.554	39.1
Total	3s. 6.07d.	84.0c.

Cost of Treatment at Modderfontein B.—November, 1914

42,000 Tons Milled

	Per Ton Milled Pence	Cents
Ore sorting and breaking.....	2.646	5.3
Transport from crusher station to mill bins.....	0.349	0.7
Stamp milling.....	8.754	17.5
Tube milling.....	6.251	12.5
Amalgamation.....	1.834	3.6
Cyaniding.....	14.531	29.0
Total.....	2s. 10.4d.	68.6c.

In closing this necessarily limited survey of Rand metallurgical practice, I must acknowledge having referred freely to those excellent sources of technical information, the *Transactions of the Chemical, Metallurgical and Mining Society of South Africa*, and the very complete *Text Book of Rand Metallurgical Practice*. I wish also to express my obligation to K. L. Graham and J. R. Thurlow of the Rand Mines, Ltd., Metallurgical Staff, and to the Mechanical Engineering Department, for assistance rendered in the preparation of this paper.

DISCUSSION

SIDNEY J. JENNINGS, New York, N. Y.—I notice Mr. Bosqui says that zinc dust can be used profitably. I had occasion recently to try to ship some zinc dust to South Africa, owing to the high price of zinc in the last six or eight months and the difficulty in getting zinc sheets. We were informed that the steamship companies in England refused to accept zinc dust for shipment through the tropics on the score that it was an explosive mixture.

C. W. MERRILL, San Francisco, Cal.—In the early days zinc dust was packed in wooden casks. The steamers at that time were very chary of transporting it. It was put on the deck in the wooden casks, and if the zinc dust got wet it exploded. Since that time we have used hermetically sealed cases, in which case there is no risk whatever in its transportation. This rule of the steamship companies is perhaps another sample of their ultra-conservativeness.

HENNEN JENNINGS, Washington, D. C.—Unfortunately, I have not yet read Mr. Bosqui's paper, but believing it might interest you to know something of the beginning of the cyanide process on the Rand, I will speak briefly as to my own knowledge and experience of the early initiation of this process in South Africa.

I took the position of consulting engineer of H. Eckstein, the parent firm of H. Eckstein & Co., and a branch of the firm of Wernher, Beit

& Co., at Johannesburg, in December, 1889, and at that time the Robinson Co. managed by them was the premier company of the fields. Up to this date plate amalgamation and catchment of a small percentage of concentrates by blankets, and in a few instances Frue vanners, were the main gold-saving devices used and had obtained an extraction of only about 65 per cent. of the gold contents of the ore. Also at that time the orebodies were seen to be changing from oxidized deposits to pyritic at varying depths from 50 ft. to 300 ft. below the surface, and uneasiness was felt as to future recovery. The deepest exploitation at any of the mines was not much below 300 ft.

Various treatments of the concentrates had been attempted by the Newbury, Vaughton, Pollock & Hungarian process, but with unsatisfactory results. Naturally I turned for further help to the best that had been done in this country in connection with the metallurgy of gold and which, at that time, was the careful concentration by vanners and treatment of concentrates by the Plattner chlorination process. Mr. Butters, at his works at Kenneth, had thoroughly established his ability and reputation in this connection and he was selected to take charge of the erection of a chlorination plant at the Robinson mine. Mr. Butters arrived about the end of 1890 and started at once on the erection of a thoroughly up-to-date chlorination plant.

In the meantime the MacArthur-Forrest Cyanide Process syndicate representatives had given public demonstrations of their process at the Salisbury Co.'s mill in June, 1890. I was privileged to make the first test of this process. Mr. MacArthur and Mr. Forrest were employed by a firm of Scotch capitalists to go through the whole literature of gold extraction and if possible to obtain a practical and efficient commercial method of gold extraction. The result was the MacArthur-Forrest process, one of the few instances of a process made to order. Mr. MacArthur in no way attempted to disguise the fact that the dissolving of gold by cyanide solution and its precipitation on zinc plates was known to chemical students. His claim was that he demonstrated the practicability of using very dilute solutions and their precipitation not on acres of zinc plates but on compact mattresses of zinc shavings. Mr. MacArthur was also fully aware at that time of the factor of fine grinding in obtaining high extraction.

The South African rights of the process were in the hands of the Gold Recovery Syndicate of which Cordner James was head. Mr. MacArthur and Alfred James came to South Africa to demonstrate and introduce it, and they were present at the first tests made at the Salisbury Co.'s plant. The demonstration plant was constructed on the agitation principle and in the test that I followed through, both concentrates and tailings from the Robinson mine were treated. I remained two days and a night at the works. We obtained actual gold buttons which showed

about all that was claimed for the extraction and, in fact, the extractions were in accord with the best subsequent practice. No information regarding costs was given, although I afterward learned it was between 20 and 30s. per ton.

As no information of costs was given, the Robinson Co. was unwilling to construct the first plant, but gave a liberal contract on a sliding scale for the syndicate to work about 10,000 tons of accumulated tailings, the Robinson Co. reserving the right to take over the plant at cost price and continue the working on a royalty basis, with a clause that no better terms were ever to be given other workers. The syndicate constructed the first working plant at the Robinson mines. At this plant, designed by Alfred James, agitators were given up and a system installed of working with square tanks and gravity percolation which contained all the vital principles of the great modern plants that have been improved and developed in details by numerous and various workers.

The syndicate put in charge of the Robinson plant George Darling, who was retained for many years as manager of the Robinson cyanide works. At first the cost was as high as 13s. per ton, but it has gradually been reduced to below 2s. In the meantime Mr. Butters had started the chlorination plant at the Robinson and had done it in such an excellent way and made such a success of the works that he was requested by the Robinson Co., of which I was consulting engineer, to design and erect a new and additional plant for cyaniding at the Robinson, in conjunction with the old one. In this plant he made use of round tanks with bottom discharges, instead of square. The diameter of the tanks, due to my caution, was kept to the maximum of metallurgical practice, which, at that time, was only 23 ft. Large cement cyanide tanks, 40 ft. diameter, were first made use of at the Langlaagte Estate & Gold Mining Co., Ltd. and were designed by J. R. Williams who afterward became the cyanide manager of the Crown Reef Gold Mining Co. of which I was also consulting engineer; at a later date he there developed the first successful treatment of slimes on a large scale by the decantation method.

In this field Mr. Butters and Mr. von Gernet were also effective workers.

The Nigel Co., of which I was consulting engineer, was the first (in 1892) to deal successfully with the problem of pyritic tailings. The success of this plant was largely due to my assistant, Fearnside Irving. The direct filling of cyanide vats was first tried in 1893 under my direction at the Salisbury mill, which had square tanks, but the advisability and success of this method was only demonstrated in 1893 and 1894 at the New Heriot mine, with round tanks, at which W. K. Betty was cyanide manager and was working under my technical direction.

The method of filling the vats at the Salisbury and Heriot was by means of a hydraulic classifier and hose distribution in connection with

slat discharges. Afterward a revolving circular distributing system was invented by Messrs. Butters and Mein and installed at the Robinson Co. Mr. Butters severed his connection with the Robinson Co. in 1892 and established a metallurgical company known as the Rand Central Ore Reduction Co. He did most excellent work in connection with this corporation and among other things gave a thorough trial to the Siemens-Halske electrical precipitation process. Mr. Betty's method of adding acetate of lead in the cyanide treatment made it hard for the Siemens-Halske process to compete with the zinc process.

It is needless to go further in the evolution of the cyanide process on the Rand, but without it few of the mines, which are now producing some 40 per cent. of the gold of the world and employing some 200,000 workers, would be in operation.

W. A. CALDECOTT, Johannesburg, South Africa.—The responsible position on the Rand occupied by the author during the past five years, his facilities for observation and experiment under working conditions, the wealth of detailed ore-treatment data at his disposal, and his association with the introduction and trial of Merrill zinc-dust precipitation, Butters vacuum filters and other appliances, entitle his opinions to close scrutiny and careful consideration. The obligation conferred, however, upon his readers, would have been even further enhanced if the expression of his opinion in general terms had been more amply supplemented by definite figures upon such crucial points as percentage recovery by amalgamation, ore-treatment costs for more than one company, assay values of residues, etc.

The paper covers so wide a field that detailed criticism of all the points raised would occupy much space, and many features of interest, upon which various opinions exist, must be passed over. The author reaffirms his belief⁴ in the general economic suitability of Rand methods to local conditions; possibly the "exhilarating atmosphere" elsewhere of numerous innovations is more congenial to the temperament of enthusiastic inventors than to the shareholders in the mines concerned. In other words, Rand practice has been developed in accordance with the axiom that "metallurgy is the art of making money out of ores," and the fiduciary position of those responsible for advising upon the expenditure of shareholders' money on reduction plants necessitates the relegation of other considerations to a secondary position.

The classification of variations in current ore treatment shown on p. 27 is of interest, but does not include the disposal of current sand residue as a pulp down bore holes for mine filling, a system which is

⁴ *Mining and Scientific Press*, vol. 108, No. 18, p. 736 (May 2, 1914).

extensively and increasingly practiced;⁵ nor is there any reference to the collection of slime, for both decantation and vacuum-filter treatment, being carried out in ordinary decantation vats; nor to the slime residue in either case being pumped as a fluid pulp to slime residue dams.

The proposed scheme (p. 29) for dealing with the 9-mesh ore from the breaking station is an interesting variant from current arrangements, and virtually constitutes a separate circuit for classifying out the solids and returning to the trommels the water in this product. Some increase in capital cost and attendance will, however, be involved by its adoption, and the same object could be more simply attained by allowing the washings to enter a pulp launder in the tube-mill circuit, the coarser solids thus added being compensated for by running some stamps with finer screening or an additional tube mill during such time, and the additional water being dealt with by adequate safety cones following the tube-mill cones.

Doubtless, owing to the author's personal experience of Rand methods only dating from 1910, some confusion as to developments prior to that date is apparent in the suggestion on p. 33 that the introduction of tube mills caused the development of heavy stamps.⁶ On the previous page the reason for the introduction of the latter is correctly stated as involving fewer crushing units with lower initial and operating costs, whereas the function of tube mills to secure cheaply any desired fineness of the crushed ore, and the coarse stamp-mill product most suitable for tube milling can be as readily obtained with a 1,250-lb. stamp as with a 2,000-lb. stamp. In passing, it can hardly be considered now (see p. 1013) that 9-mesh (0.272-in. aperture) is the economic limit of screen mesh, as 0.375-in. and 0.5-in. apertures are in regular and satisfactory use at the large Robinson Deep mill and up to an inch aperture has been successfully employed elsewhere.⁷ The reason why these last screens are not in more common use is that they permit a comparatively small increase in stamp duty above a $\frac{3}{8}$ -mesh aperture, and are liable to cause banking in existing launders of limited grade.

⁵ *Journal of the Chemical, Metallurgical and Mining Society of South Africa*, vol. xiv, p. 119 (Sept., 1913).

⁶ Compare remarks in Presidential Address by Prof. G. H. Stanley in *South African Journal of Science*, p. 46, November, 1914, as follows: "Tube mills were originally installed really to grind 'concentrates' from spitzluten to improve extraction; this so shortened time of treatment that separate treatment became unnecessary. Finer grinding was then adopted for all the ore; this led to coarsening the battery product, this in turn to abandonment of front plate amalgamation and the recognition of the stamp battery as a crushing rather than an amalgamating appliance, which at one time was considered to be par excellence. Next, in the effort to improve the output, the weight was increased, and percentage of water used decreased with consequent lowering of pulp elevating charges."

⁷ H. A. White, *Transactions of the Institution of Mining and Metallurgy*, vol. xxii, p. 185 (1912-1913).

The statement on p. 40 that in future at the Modder B a tube mill will deal with 264 tons per day of 9-mesh product mainly from ordinary stamps, whereas it can only handle 203 tons per day from Nissen stamps, indicates, assuming similar final pulp, that the screen pulp in the former case is much finer than in the latter, thus throwing less work upon the tube mill. Since the same battery screen is used in both cases, this illustrates the well-known fact that the stamp duty is no criterion of the work done in the mortar box, unless account is taken of the grading analysis of the screen pulp. The assumption on pp. 1006 and 1009 that 1,650 lb., and not 2,000 lb., is the economic limit of ordinary stamp weight is possibly based on trouble experienced through cam-shaft breakages when the latter were first employed with too high a number of drops per minute, but the fact that 1,900-lb. to 2,000-lb. stamps have been installed in nearly all recent mills (Consolidated Langlaagte, Government Mining Areas, Van Ryn Deep, Brakpan Mines, Roodepoort United, Main Reef and Modder Deep) shows that this difficulty has been overcome.

The flow sheet of a proposed mill on p. 36 shows a double row of stamps, which is undesirable with heavy coarse-crushing stamps, both on account of the relatively limited bin capacity with high-stamp duties, and likewise because of the excessive lowering of the tube-mill cones required for the pulp from the back row of stamps. This loss of fall, involving additional pulp re-elevation cost, is accentuated by placing the cones of four tube mills low enough to receive pulp interchangeably, whereas three constitute an ample measure of interchangeability. As the diagram shows, the tube-mill cones are at a very low level, and the tube mills, plates, and return pumps consequently require a considerable depth of excavation.

In reference to the author's discussion of variants of amalgamation methods, the essential point appears to be missed, viz., that in modern mills the crucial factors of amalgamation recovery are the degree of liberation of the gold by the extent to which crushing is carried, and the efficiency of the pulp classification. The area of plates in the tube-mill circuit is quite secondary to these and mainly dependent upon the volume of pulp amalgamated, since with efficient classifiers gold particles of appreciable size merely circulate until amalgamated. The flow sheet of the proposed plant given in the paper shows that the overflow of the tube-mill safety cones joins the tube-mill outflow. With an imperfect arrangement of classifiers this may be necessary, and the cost of a considerable re-elevation of many thousand tons of overflow pulp daily in a large mill may be a warrantable expenditure in order to amalgamate gold particles overflowing the safety cones. The increased volume of pulp thus passing over amalgamated plates might require five of the latter per tube mill as advocated by the author, but when the tube-mill outflow only is amal-

gamated, three plates per tube mill have proven ample in practice under varying conditions on different mines, and at the Robinson Deep during 1914 with the design illustrated by Diagram C on p. 45, the amalgamation recovery has amounted to 65.23 per cent. of the original gold contents of the ore, the sand residue averaging 0.315 dwt. per ton in value without ever showing any trace of free gold. It may be added that during 1914 this plant treated 576,370 tons of ore with a total residue of 0.252 dwt. per ton, equivalent to a recovery from all sources of 96.32 per cent., the slime original being 1.802 dwt. per ton and its residue after decantation treatment without air-lift agitation being 0.181 dwt. per ton. The appearance of free gold in the sand residue at the Modder B may be due to various causes, but so far as classification is concerned one small safety cone between two tube-mill cones is liable to considerable overloading with pulp if its feeding cones are overloaded, and the installation of return cones shown on p. 1022 for receiving tube-mill return pulp does not appear to be provided with safety cones, so that irregular distribution at this point is likewise liable to result in oversize particles or amalgam overflowing. On the other hand, the simple classification system shown in C (p. 45), whereby tube-mill and safety cones only are required and a minimum cost of pulp elevation is involved, permits of irregular or fluctuating distribution of pulp to the former to be corrected by safety cones of ample size receiving the total overflow of the tube-mill cones from a common launder.

In emphasizing the utility of Schmitt tube-mill feeders, the author expresses the common opinion of users but does not refer to the advantages of pebble bins with automatic pebble feeders actuated by the tube mill. He likewise confirms the advantages of the scoop discharge for tube mills as recently described in detail by W. R. Dowling⁸ and refers to the desirable location of tube mills below stamps as illustrated by the same metallurgist over four years ago⁹ in the form shown in Diagram C.

As the scheme for a proposed new reduction works illustrated in Fig. 7 is not in actual operation, comment thereon can only be based on past experience of the various devices shown and on their relation to each other; it is to be regretted that the author had not the opportunity of demonstrating by actual results the advantages or otherwise of his proposed plant as compared with existing practice.

The undesirability of a double row of heavy stamps and the excessive volume of pulp to be amalgamated and reëlevated has already been referred to, as well as the absence of safety cones following the return

⁸ *Journal of Chemical Metallurgical and Mining Society of South Africa*, vol. xv, p. 214 (March, 1915).

⁹ *Journal of Chemical Metallurgical and Mining Society of South Africa*, vol. xi, p. 421 (March, 1911).

cones. The elevated position of these last further involves much coarse pulp being raised to them merely to gravitate down to the tube-mill cones, instead of being elevated only to the height of the latter, according to the method shown in Fig. C on p. 45.

The Dorr thickener shown, while a useful device when crushing with cyanide solution for removing surplus solution from slime pulp prior to air-lift agitation or vacuum filtration, is not adapted to effect as complete a separation of water from settled slime as the ordinary intermittent settlement and decantation. The author's assumption that the latter yields a settled slime containing 50 per cent. of moisture with banket ore slime, which is a very good result for a continuous thickener, is apparently based on data from obsolete and inadequate plants, since in a modern slime plant 40 per cent. of moisture in settled slime is by no means the highest degree of settlement obtained under ordinary working conditions. This difference of moisture content is of great importance, because with a 50 per cent. settlement half as much water again is added to the working solution as with a 40 per cent. settlement, and in consequence the same additional amount of dilute working solution has to be regularly discharged with its gold, cyanide, and lime contents.

As regards the vacuum filter, the author places the extraction by this process at 93 per cent. This is appreciably higher than by decantation, but is obtained at an extra cost of $2\frac{1}{2}$ d. to 4d. per ton. Comparing this with the Robinson Deep figures quoted earlier by the writer, which show an extraction by decantation slime treatment of practically 90 per cent., it will be seen that the difference of 3 per cent., equal to 2.7 d. on 1.802 dwt. slime, is slightly less than the average extra cost of filtration. Hence the author's dictum that vacuum filtration should be adopted in all new plants may be received with some qualification, and as being specially applicable to rich slime where a small additional percentage recovery on the value is profitable.

The inference on p. 51 that thorough separation of sand and slime by classification is due to the influence of vacuum filtration is hardly warranted by present or recent practice, since the achievement of such separation has been in vogue prior to 1910,¹⁰ on plants where decantation practice obtained and is still practiced. In fact, at the present time the most advanced practice is to confine the product separated as slime to colloid particles so far as possible, since fine granular particles of silica or pyrite yield a very good extraction when included in the sand charges without interfering with leaching, whereas their presence in slime charges delays dissolution of its gold contents, raises the value of slime solution, and increases wear on slime sludge pumps.

Fig. 7 does not indicate the disposal of the water drainage from the

¹⁰ *Journal of Chemical Metallurgical and Mining Society of South Africa*, vol. x, p. 51 (August, 1909).

sand vat when filled, but the proposed treatment in this vat of the sand collected from a water-borne pulp is a revival with cleaner sand of Rand practice of 15 or 20 years ago.¹¹ As the author states on p. 52, it was formerly customary to apply weak cyanide solution in the sand collectors, but he omits to give the reason for the abandonment of this practice, which was the contamination of the mill service water with cyanide and consequent dissolving and loss of gold contained in the ore crushed. The seriousness of this loss may be gauged not only from the above fact, but from published references,¹² of which possibly the most notable are the following, *Transactions of the Institution of Mining and Metallurgy*, vol. xiii, pp. 77 to 84, Nov. 19, 1903, by Charles Butters and Hennen Jennings. Charles Butters said:

No cyanide solution is ever run into the collecting tank (*i.e.*, when equipped with a Blaisdell excavator) as its contents, after having been filled from the battery, are allowed to drain, and are then transferred by means of the excavator direct to the leaching tank. Therefore no loss of gold can take place from cyanide solution having been introduced into the collecting tank, as is the case when the collecting tank is filled directly from the battery, and gets its preliminary treatment of cyanide solution in the same tank. I have yet failed to find a reservoir on the Rand that did not have gold in solution from this cause. This gold in the reservoir varies in value with the season—being higher at the end of a dry season than at the commencement of it, because during the dry season the reservoirs never overflow, but become concentrated from evaporation in the warm atmosphere.

At Virginia City we found as much as 50 per cent. of gold dissolved out of the charge, by filling into a collecting tank in which the charge before had received a preliminary cyanide treatment, and, strange to say, this loss continued for over two weeks after we had discontinued the use of cyanide in the filling of the vat, and this, notwithstanding that we had washed it out as clean as we could. While we could obtain no trace of cyanide at any time, still the loss went on, until after two very anxious weeks it gradually stopped. This occurred on two different occasions. At first I attributed it to an excessive amount of ferric salts in solution, acting possibly as a solvent for gold; but we found out later that it was from the traces of cyanide that could not possibly be detected by any chemical test.

Mr. Hennen Jennings and myself had both made a series of determinations at Johannesburg as to the gold in the battery water from this source, and with similar results—gold was invariably found. It is not easy to state the exact loss from this source, but that it is a source of loss which should be eliminated I think all engineers and metallurgists will agree.

Hennen Jennings said:

The remark of Mr. Butters as to gold being found in the battery water was correct. He regretted that he had no detailed figures with him. Some very extensive tests had been made at mines with which he had been connected, and it had been found

¹¹ F. C. Pengilly, *Transactions of the Institution of Mining and Metallurgy*, vol. vi, p. 113 (Jan. 19, 1898).

¹² A. von Gernet, *Journal Chemical Metallurgical and Mining Society of South Africa*, vol. ii, p. 529 (January, 1899); and H. T. Pitt, vol. v, p. 84 (September, 1904).

that even in large storage reservoirs the water held a slight amount of gold in solution. It varied from a trace up to sometimes as high as 1 dwt., but that was very exceptional.

That a good extraction from clean sand can be obtained by cyanide treatment of sand in the vat in which it is deposited from a solution-borne pulp is evident from the results with this method which is in present operation on the Rand at several mines.¹³ This fact, however, does not dispose of the point that actual recovery of the gold and not merely extraction from the ore is the true objective, and the author does not disclose how he proposes to overcome the contamination of mill service water circuit and the consequent loss of gold, resulting from alternately introducing into the same vat and saturating the filter cloth with water and cyanide solution. Roughly, the consumption of water in a Rand reduction plant is half a fluid ton per ton of ore milled, and this loss mainly occurs as moisture in sand and slime residues. In addition, however, water is consumed by mine-filling with sand residue pulp, by the periodical discharge of slime residue with mill service water, by liability to seepage and overflows at some point in its circuit, and by incidental use for many minor purposes; in general, long experience has emphasized the desirability of maintaining the mill service water as free from gold as possible, as stated by the authorities quoted above, if mysterious shortages of gold called for are to be avoided. The maintenance of free protective alkali in the form of calcium hydrate in mill service water on the Rand prevents the rapid decomposition of traces of cyanide by acid iron salts in the ore, and since sodium cyanide is capable of dissolving about twice its weight of gold, the presence of only 0.0001 per cent. of sodium cyanide in mill service water may be equivalent to over a pennyweight of dissolved gold per ton of water.

A. J. CLARK, Lead, S. D.—One point I think ought to be made is this: That the establishment of the decantation process creates a large volume of low-strength solution which is successfully precipitated by zinc shavings, where zinc dust has made only a qualified success. I think this point needs emphasis, that zinc shavings would not precipitate these solutions for any length of time. New shavings are used for a very few days and then the shavings are transferred to the box in which the original solutions of the sand plant are treated. The process in Africa thus conforms to the requirements of zinc-shavings precipitation, but the zinc-dust process has to stand by itself in each instance. The zinc dust cannot be used again. It is not possible to take it from the press and use it in a stronger solution successfully. At least I have never found it so.

¹³ *Transactions of the Institution of Mining and Metallurgy*, vol. xxii, p. 182 (1912-1913), (which shows a sand residue of 0.233 dwt. per ton by single treatment in the collecting vat).

C. W. MERRILL, San Francisco, Cal.—That is an interesting point. It shows how the process is developed according to local conditions. In Africa they have a mild climate and cheap labor, but up at the Homestake we have a very severe climate and expensive labor. We developed methods whereby we were able to handle larger bulks of pulp and still not exceed the cost factor. Local conditions will to a certain extent determine the trend of the process.

G. H. STANLEY, Johannesburg, South Africa (communication to the Secretary*).—It might be well to emphasize the great improvement which modern results in cyaniding show in comparison with easy practice as deduced, for instance, from examination of old dumps.

In several cases the interiors of these have been laid open to examination as a result of sand-filling operations by hydraulicking, etc., and assays have shown the presence of thousands of tons assaying as much as 3 dwt. per ton gold in the older and bottom portions, while even the newer portions, though still made prior to tube-milling days, give assays frequently approaching 1 dwt. and in some cases considerably in excess of that figure.

One dump of about 1,500,000 tons averaged 1.2 dwt., another of over 1,000,000 tons averaged 1.65 dwt., and these values have probably been reduced from the original by years of atmospheric leaching in the presence of thiocyanate in the acid dump. This high value was undoubtedly due in great measure to the coarseness of crushing which obtained, grading tests showing the presence of 35 to 45 per cent. + 60 sand particles, but at the same time a large proportion of slime is present, often in distinct layers, which prevented proper treatment of the sand and itself received practically no treatment on account of impermeability.

Slime washed from the sand and assayed showed values generally about twice as high as the average of the dump. The washing was consequently very imperfect as is evidenced by the frequent presence of patches of Prussian blue color.

These points are, of course, remedied in modern practice by fine grinding to practically all through 60 mesh and by better classification, so that at present a good sand residue may assay only 0.25 dwt. Of course, the original value of the sand treated was formerly much higher than obtains at present and the extraction percentage was apparently fairly satisfactory, but that the high residue value was not due simply to the high original value is shown by the fact that a residue value which would be considered extremely good in current practice can be obtained by regrinding and retreatment of this old residue. The retreatment of these old residues is, however, in most cases precluded by their being covered up by later

* Received Sept. 2, 1915.

low-grade residues; by their great acidity, necessitating the use of excessive amounts of lime; and other causes.

Sand dumps are regarded as a necessary evil, and give rise to much discomfort by the dissemination of dust in windy weather. Various means have been tried for preventing this, the most successful depending upon binding the particles together by spraying the outer surface of the dump with black veil mud or ore slime and salt mixed with water, both these methods being now in effective use on one or two favored properties.

There was obviously in the early days every incentive to treat with the sand as large a proportion of slime as possible, otherwise it received no treatment at all. Consequently the introduction of the decantation process by Mr. Williams constituted a great advance indeed. Before making my first personal acquaintance with the method (in 1905) I was well aware of the theoretical imperfections of the method and had busied myself with the development of a suction filter. It did not require a very long while of actual contact though to convince me that my suction-filter idea (on much the same lines as the present Moore or Butters but at that time unsupported by experience) had little chance of adoption in place of the very cheap and effective operation of the decantation plants. The margin for improvement in saving was apparently so small; and, although in the few years following much experience was gained elsewhere, only a very confident technical adviser and very strong financial interests could afford to take the respective risks involved. I am very glad that the introduction of suction filters has been so well justified.

Nevertheless there seems to be no prospect of the whole of the ore being treated as one product; local feeling can be summed up in the statement that while we know slime cannot be treated like sand, we hold that sand is best treated as sand and not as slime.

As with suction filtration, so with precipitation, and I am in complete agreement with Mr. Bosqui in his opinion that this is the department in which the most notable advances can and will be made. The present method has always appeared to me to be the weakest part of the whole sequence of operations and I am confident that if the same amount and quality of attention had been devoted to electrical precipitation as to zinc precipitation, a better method would have been evolved. Of the two, the zinc method appears more understandable to the practical worker deficient in scientific training and the variable factors in its operation are more or less within his ken, so that he can easily experimentally vary the conditions and ascertain those most suitable, while in the electrolytic method expert scientific knowledge and appreciation of the effects of varying factors are much more urgently required.

Experimental work in this direction has been carried out by graduate students in my laboratory and I understand that their work is being extended on one of the mines. I am certainly hopeful of the results.

Breaking and Sorting.—Sorting and breaking are commonly done at the shaft head in the “crusher station,” and the broken ore is then transported to the mill, either in hopper-bottom cars, often of full railroad size, drawn by steam or electric locomotives running over the battery bins, or alternatively conveying and elevating belts are used.

The non-crystallization of design into some standard method is partly explained by the difference in the amounts of waste sorted out at different mines, which in turn depends considerably upon the variation in thickness and dip of the reefs. This variation affects the relative amounts of rock broken by machines or hammer boys and hence the amount of waste mined.

Undoubtedly the use of circular sorting tables in buildings with several superposed floors gives rise to crowding in a breaker station; while with belts the building is spread out over much more ground space, there is more room available, and better lighting is possible, which is in favor of the employment of belts.

I must confess to a feeling of preference for jaw breakers as opposed to gyratory crushers, mainly on the ground of greater ease in clearing after a choke or sudden stoppage, and of renewal, replacement, or repair. Mr. Bosqui does not give the size to which the ore is broken for stamp milling, and although about 2 in. is aimed at, the feed in most cases is actually nearer 4 in. Jaw breakers for this size are satisfactory and the larger size is apparently of no moment with modern heavy stamps.

Stamp Milling.—Here the Rand has little to reproach itself with on the score of conservatism. The figures given relative to the superior performance of Nissen stamps appear to be conclusive.

Amalgamation.—Mr. Bosqui appears to cavil at the extreme to which reduction in the number of tube-mill plates has been carried on the Gold-fields group. This of course is accompanied by efficient classification so that such portion of the pulp as is bypassed to the cyanide plant will only carry a little free gold and that in a finely divided state; so that in the first place it would only be amalgamated with difficulty on account of its slow settlement and in the second it is easily soluble in cyanide.

It is commonly stated that gold amalgamated represents 100 per cent. extraction, while if it gets to the cyanide plant only 90 per cent. can be recovered, and this is one of the half-truths so difficult to controvert. The residue from cyaniding contains gold partly undissolved (because incased, etc.) and partly as dissolved gold which has not been washed out, the former being much the more important in amount. The finely divided free gold referred to, which presumably might have been amalgamated, will presumably also be completely dissolved in cyanide and nearly all, not 90 per cent. only, removed and recovered; the value in the residue is mainly due, as stated, to incased gold which could not have been amalgamated either. There is, of course, no need

to point this out to the author, but it may not be obvious to some of his readers.

Practically all the free gold which would be able to settle in the stream of pulp passing over the plates and so become amalgamated, settles in the classifier and therefore gets on to the plate, as is shown by the very small difference in recovery when bypassing and using few plates, or when using the alternative practice, in each case under good conditions.

In connection with the trials at Modder B, we are not told whether any pulp was bypassed and I suggest that the appearance of free gold in the residue indicated merely overcrowding of the plates with pulp and that a cure might have been found in bypassing some, or more, of the pulp—the carefully classified finer or lighter portion, of course—and should like to know whether this was tried. This view obtains support, I think, from the fact that overcrowding of the full 30 plates also led to the appearance of fine gold in the residue.

Tube Milling.—Mr. Bosqui on p. 41 refers to the broken quartz necessary to maintain the pebble load. I merely mention this as it might give a wrong impression—it should, of course, read, ore or banket.

With regard to peripheral discharge, similar experiments to those instanced at the City Deep have been in progress elsewhere and the results are given by Mr. Dowling.¹⁴ As a result the wide adoption of scoop discharges seems to be foreshadowed, but will necessitate the provision of increased power. Mr. White has also put on record before the same Society results of experimental work with a short section of a tube mill of variable diameter, which I had the pleasure of superintending.

In the case of the tube mill with ball mill end described, it would seem that the fact that partitioning off the end of the mill in that manner entails no decrease in efficiency constitutes an argument in favor of the view that the length of mills might be lessened. The few balls would have little action on the already fine pulp, certainly much less than the usual charge of pebbles; their action would be practically confined to the small pebbles to be crushed.

Classification.—The author's conception of the mode of action of a diaphragm classifying cone appears to be quite exceptional and I hesitate to accept it. I should like to ask whether he has any evidence that the outer portion of the settled sand is crowded upward? The central portion is certainly coarser and therefore heavier so far as the weight of individual particles is concerned; but I have often noticed that the outer and finer portion appears to be much richer in pyrite and consequently the specific gravity may be even higher than in the center.

¹⁴ *Journal of the Chemical, Metallurgical and Mining Society of South Africa*, vol. xi, p. 414 (March, 1911).

Unfortunately, I have no figures available. My impression of the mode of action is that the concave shape of the top surface is determined by the fact that the disturbance at the central feed is too great to allow any but the very coarsest pieces to settle; as the pulp flows out toward the periphery the velocity decreases so that more and more sand of successively finer grade settles out, the feed being so regulated that only particles sufficiently fine reach and are carried over the edge, but that all the settled sand is slowly settling toward the outlet. The coarse center tends to move faster than the sides and this would lead to "break-aways" if it were not checked by the diaphragm which serves both to take pressure off the outlet and to mix the sand in passing by its edge.

The separation of sand and slime is more difficult than the preparation of tube-mill feed, as the particles are more nearly equal in size. Where the same kind of cones are used, the underflow (sand) is considerably contaminated with slime, and is diluted with wash-water and reclassified, using in each case a thick underflow; or else, as the author describes, a separate clean water service is fed to the bottom of the cone, inside, which prevents slime issuing with the underflow sand—washes it, in fact—but then uses a thin discharge; in either case the final sand contains very little slime, and it therefore becomes possible to add the required lime in the battery as the difficulty of premature slime settlement in sand is thus obviated.

Treatment of Sand.—There must be few plants operating as described under method 1. It is usual, where hose filling is used, to pass the mill pulp through spitzkästen or spitzluten, and convey the spitz underflow, still containing much slime, to the collectors through hose. It might be mentioned, too, that at some plants a considerable proportion of the residue is disposed of by sending it immediately underground for stope filling.

In connection with the filter-table method of sand collection, the sand is ploughed off into cyanide solution and then pumped to collecting vats via Butters' distributors; it is consequently agitated and collected in cyanide solution. Dissolution of the gold is thereby facilitated, and treatment time decreased, necessitating less plant and consequently economizing capital. In this connection it may be mentioned that for a given daily capacity the cost of a reduction plant is now only about one-third of that before the war.

Slime Treatment.—I have already referred to the decantation method and would only add now that the chief trouble met with is unsatisfactory settlement, particularly in cold weather, and on one or two plants this has been overcome by Mr. Salkinson's method of warming the pulp by the utilization of waste steam from the hoisting or mill engine, this incidentally increasing the capacity by as much as 20 per cent. Un-

fortunately the general use of purchased electric power makes this impossible on new plants.

With regard to air agitation it should be pointed out that the use of the usual tall, narrow Brown tanks is not universal. Alternatives are the employment of tanks about 30 by 30 ft., in which the central air lift does not extend to the top, and which it is claimed use therefore less power; and in some very large vats there are several comparatively narrow air-lift tubes, with tangential discharge, so as to set up a swirling motion of the whole pulp in the vat, and prevent too rapid settlement.

In connection with the Usher process it was understood that the vat acted as a huge classifier, so that the charge was not homogeneous and correct sampling became almost impossible.

Smelting.—From either method the slags still carry several ounces of gold per ton, even after crushing and panning, and even on plants where “pot” smelting is employed a pan furnace is often installed for smelting these slags with lead, in order to recover this gold; at the same time any other gold-bearing byproduct about the plant may be similarly dealt with for the extraction of its gold.

Even after this, the slags obtained may receive a further treatment by smelting in a blast furnace, two or three plants having their own, but more usually selling it to a byproduct works.

There were formerly several of such works, operated by independent people, but a coöperative byproduct treatment works has also been established by the gold-mining companies, to which those companies which do not possess their own facilities dispose of the byproducts, which are there treated as at the larger mines.

In conclusion, while there is uniformity in the main outlines of practice, it will be evident that many variations in detail exist among the 50 producing mines.

This is not due entirely to the idiosyncrasies of controlling officials, for the ore itself, and other circumstances, vary somewhat from mine to mine, apart from variations in value. In some places it is harder than in others; again, the nature of the country rock exercises an influence, sericitic schist, for example, adversely affecting amalgamation, and in some cases a small proportion of other metallic minerals occurs in the ore, and doubtless has some influence; I have, for instance, found traces of telluride in some cases.

It will be seen that no concentration, either “clean” by tables for subsequent chlorination, or simply rough for separating part of the pulp for longer cyanide treatment, is now employed, and, in consequence, chlorination has been completely discontinued.

There is, however, a small proportion of very clean and rich pyrite collected chiefly with amalgam, after separation from amalgam and grinding finely in small tube mills, etc., which is treated simply by

agitating by air in a special small cyanide agitating plant—as first suggested by A. F. Crosse—and a large percentage of the gold is thus extracted and dealt with as usual, the residue commonly going to the main cyanide plant.

This pyrite contains appreciable amounts of osmiridium, which may possibly be turned to account in the future.

W. R. DOWLING, Germiston, South Africa (communication to the Secretary*).—Under "Classification" on p. 44 the author is somewhat in error in describing the earlier classification methods. The inverted pyramidal spitzkasten were used merely to remove some of the water in the tailing pulp, although incidentally some slime was also carried over, and the underflow being reduced in volume was more easily handled in the sand collector vats where the main separation of sand from slime took place.

On pp. 46 and 50 the author refers to the use of the vacuum sand filter table as being a method for obtaining clean sand. Obviously this cannot be the case, as the filter table merely dewateres the clean sand previously obtained by cone classification. The sand being thus freed from water is then ready for cyanide treatment, without fear of complications arising through the introduction of cyanide solution into the mill water. The author when discussing the separation of sand from slime somewhat confuses the question by introducing the classification of pulp for tube milling into the same paragraph. When analyzed, the statement in question appears to mean that the separation of sand from slime was to be effected in large cones in the same way as with those already in use with sand filter tables, except for the use of water at the apex of the cones in the former case. For its proper treatment, sand should in any case be free from slime, to the extent at least that layers and lumps do not exist, but the author is in error in holding that "an imperfect knowledge of classification, or a failure to recognize its importance, made this scheme (i.e., the treatment of sand in the collecting vat) impracticable." That the classification in the light of later knowledge was imperfect is true, but an equally important cause for the abandonment of sand treatment in the same vat as collected was the fact that gold in solution was lost. Quite apart from any accidents due to solution being pumped on to a vat with leaching pipe or overflow doors open to water storage vats, or similar accidents by which gold-bearing cyanide solution entered the mill water, there were the difficulties of removing the gold-bearing cyanide moisture from the inch or so of sand usually left in the filter bottom when a vat is emptied of sand. The cocoanut matting and jute forming the filtering medium were similarly saturated, while the

* Received Oct. 7, 1915.

bottoms of steel vats are irregular and a series of solution pools formed. It was only when regular and systematic samplings and assays of mill water were made that these losses of gold were appreciated. I well remember my attempts, in 1899, to remove gold-bearing cyanide solution from the filter bottoms of collector vats by repeated small water washes. Although the washing reduced the loss appreciably, the extra labor and expense were considerable. It was impossible to absorb these washings into the plant as the moisture of discharged residue was the same as collected sand, so the liquid after precipitation was run to waste with consequent loss of water. This solution, by the way, was rendered acid before precipitation on iron or zinc, and the gold recovered in a precipitation plant set aside for the purpose. The mill service water, on the plant referred to, as in most earlier and some existing plants, was stored in earthen dams, and the loss by seepage of gold-bearing water must have been considerable. Although various precautions may be taken against the loss of mill water, such as the use only of vats for storage, and against the introduction of gold-bearing cyanide solution into the circuit water, these only minimize the evil and the fact still remains that the milling will be done in a dilute cyanide solution. The question then naturally arises as to what is the real objection to milling in cyanide solution. I have searched in vain for a pronouncement by the author on this matter, where he is treating an ore yielding so high a proportion of its gold contents by amalgamation and which gold, as he rightly points out on p. 37, is only commercially recoverable on plates.

The real objection to milling in cyanide solution on the Witwatersrand is that since the ore yields so high a recovery by amalgamation, indicating relative coarseness, the treatment of the ore by cyanide only would naturally be prolonged, expensive and imperfect. Should amalgamation be carried out nevertheless, complications due to corrosion of copper plates and dissolution of mercury arise. Both of these metals are subsequently precipitated on the zinc in the extractor boxes and result in base bullion, due to copper. Furthermore, crushing in cyanide solution renders it impossible to obtain accurate screen samples by reason of the presence of dissolved gold; and the cyanide goes on dissolving gold from the sample itself. It is an interesting fact that although a few mines did construct plants to mill in cyanide solution, one of the plants reverted to water, and no responsible engineer or metallurgist has since been bold enough to advise his principals to adopt this method.

H. A. WHITE, Springs, Transvaal (communication to the Secretary*).
—It is, as the author points out in his valuable paper, a fact, that in many plants the "trommel washings" are allowed to become eyesores, but the simpler system of pumping the entire product direct to the tube-

* Received Nov. 17, 1915.

mill circuit will avoid the extra expense of separation and small-scale treatment when one considers the relative distance of the points and the proportionately small amount of water.

It is a matter of record that in stamp batteries the increase of weight has been developed concurrently with the increased use of tube mills, but independently and on its own merits. The satisfactory duty, excellent running time, and freedom from breakages in the newer mills, such as the Modder Deep, with 2,000-lb. stamps, will probably go far in removing some of the conservative prejudices mentioned by Mr. Bosqui.

The success of the City Deep trial of the Nissen stamp cannot be entirely attributed to better feed distribution and discharge effects (published experiments on two-face discharge mortar boxes showed no advantages). Attention is directed to the treatment costs given by the author both before and after the increased capacity resulting from the addition of 16 Nissen stamps at Modder B. It will be observed that the only item of cost resisting the tendency to reduction caused by increased tonnage is that of milling, which has risen 1.3c.

With reference to the provision for amalgamation in the new plans, it seems obvious that a great loss of head would be eliminated by passing the tube cone overflow over a separate top series of plates; three at the top and three at the bottom will effect the maximum amalgamation. This plan is followed at the Princess Estate and Gold Mining Co., Ltd., Modderfontein Deep Levels, Ltd., and Geduld Proprietary Mines, Ltd., but elsewhere reliance is placed upon ample cone capacity and the underflow only is passed over three plates after going through the tube mill, while the overflow is passed direct to the cyanide works; both methods avoid the largely unnecessary elevation of considerable quantities of water, fine sand and slime involved in Mr. Bosqui's scheme.

The most economical use of the tube mill is still receiving considerable attention though many points have been eliminated from the field of controversy. The running speed favored by Mr. Bosqui (28 r.p.m. for the 5 ft. 6-in. tube) is very close to the point of maximum production of — 90 grade per horsepower-hour consumed. There is, however, another interesting maximum, which is that of capacity or greatest production of — 90 material per tube. This speed, 31 to 32 r.p.m., is the more likely to be adopted as it involves but little waste of power, which is, after all, not the only item of cost.

The latest Osborne bar liner coming largely into use has the upright bar held in position on top of instead of between the flat bars, as illustrated in this paper. This allows longer life for the more expensive radial steel bars and the reduction in crevice capacity is reflected in the saving of amalgam caught therein. We are still looking for the ideal liner which must be cheap and durable, have a good grip on the pebble

load, be easily and quickly renewed, and cause no locking up of gold in the form of amalgam, in crevices.

The tube-mill scoop has now passed the experimental stage and is giving satisfaction, but it would be interesting to know whence the author derives the "sloping line of discharge" from inlet to scoop. The level of pulp is surely as horizontal as in the old tubes, though the height is considerably reduced. The "whole hog" peripheral discharge had only one defender at the early date referred to.¹⁵

The scheme presented for collection and treatment of sand in fulfilment of the author's undertaking to "evolve a simpler method than the filter table" suffers from even graver disadvantages than the other well-known efforts to eliminate the table itself (the patented feature) from the whole system used under that title.

Mr. Bosqui's idea of treating a charge with cyanide solution in the same vat in which it is collected by water, re-introduces the identical defect which finally caused the abandonment of double treatment on the Rand, referred to by Mr. Bosqui. In that case also, the consequent contamination of the mill water by cyanide, and the inevitable, continuous loss of gold was aggravated by the occasional accidental heavy losses caused by allowing the cyanide solution to run into a vat still being filled by water-borne sand. These losses were doubtless better known than advertised; nevertheless the re-introduction of such possibilities is a retrograde step in defiance of the more modern practice which encourages only one method of getting cyanide solution into the mill water-stream, *i.e.*, with a bucket. The new plants of the Barnato group now working, such as the Van Ryn Deep, Ltd., keep the collecting and treatment vats separate and double wash the sand to free it from colloidal slime, getting a better effect from the added water than can be possible in the single-stage hydraulic separation advocated in this paper.

Why this conservatism with regard to the sand filter table? True, it is a local invention, yet this method is coming into extended use, and experience on mines such as the East Rand, the Princess Estate, the Geduld Proprietary, the Modder Deep, and the City & Suburban Mining & Estate Co., Ltd., warrants a belief in its future. At the Geduld and Princess the conversion of old plants enabled a true comparison to be made and the improvement in sand residues in strictly comparative experiments on a working scale of 0.05 dw't. at the Princess was much more than confirmed at the Geduld, where there was a greater margin to work on.

With the sand filter table system the dewatered sand is pumped in a stream of weak cyanide solution to the collecting vat where the

¹⁵ *Journal of Chemical, Metallurgical and Mining Society of South Africa*, vol. vii, p. 371 (1906-1907).

treatment is also finished and every particle of sand must come into contact with cyanide solution, which is not inevitably the case with any other method. Thus we get the advantages of a preliminary "weak wash" which formerly had to be discontinued because of the time lost, while now the sand is under treatment half an hour after it has left the tube-mill plates.

As far as capital cost is concerned, even with treatment in collecting vats, no great saving can possibly be shown if it be recollected that a longer time of treatment will of necessity be required with a water-collected charge, so that more vats must be installed than are required in the safer filter table method. As regards cost of maintenance and operation, a reference may be made to an extract from the official report on the filter table at the City Suburban.¹⁶ The excellent results there shown with regard to light cost and increased extraction serve to confirm the general experience and justify the adoption of the sand filter table system on the fine new plant at the Modderfontein Deep, the most recent on these fields. In this last instance, of course, no comparison with older systems is possible, but the results obtained were better than anticipated from the preliminary small-scale tests made.

The Dorr collecting vat for slime may have the attributed disadvantage of not being a local engineering product, but so far the real reasons for its absence on modern Rand plants are as follows: (1) The increased capacity for a given area is small, as pointed out by Mr. Bosqui. (2) The extra moisture contained in the underflow involves a loss of about 2c. per ton for cyanide alone (required to bring this extra water up to treatment strength). The figure given by the author for moisture in settled slime in the ordinary collector (50 per cent.) is much too high, and modern plants easily secure a reduction to 40 per cent., and still lower figures are obtained when much very fine sand is present. The use of 70-ft. vats in place of the 56-ft. mentioned is also becoming general in pursuit of the successful "large unit" policy. (3) The increased capital and running costs of the Dorr vat have insufficient advantages over the simple method now in use.

The Butters filter system, for the introduction of which our thanks are largely due to Mr. Bosqui, is highly appreciated and would have much more extended use if the slime here were generally of higher value than it is. The extra cost is balanced only by the higher extraction when the original slime value is in the neighborhood of 2 dwt. Take for example the Princess Estate where the average value of the residue is under 10c. and many other mines where it is under 20c., the undissolved gold accounting for more than half of this. In such cases it is clear that the quoted running cost of 6c. to 8c. will leave little margin to pay for the extra

¹⁶ *South African Mining Journal*, vol. ix, Part II, p. 739 (Jan. 27, 1912).

capital cost involved. This process is very successful at the Geduld with slime of initial value of 3.5 dwt. and where, by the way, the only continuous use of air-lift vats on *current* slime is as yet practiced, beside the new plant at the Modder Deep. The East Rand Proprietary Mine vats are used for accumulated slime at present.

The economy in zinc consumption claimed for the Merrill press is not warranted by the figures quoted (0.152 and 0.154 lb. per ton of solution). The average figure for the Rand is about 0.35 lb. per ton of pulp and on some plants 0.25 lb. is maintained. This worked out on a $2\frac{1}{2}$ to 1 pulp, the usual ratio, would be 0.14 lb. and 0.10 lb., respectively, per ton of solution. Any economy per ton of pulp shown by the Merrill system must therefore be offset by reduced treatment, and the figures given for gold value of the residual solutions, being distinctly poorer than those obtained with zinc shaving, contribute to the impression that this process, though not without valuable features, will have to seek a welcome elsewhere.

While it must not be overlooked that the presence of zinc increases the stability of cyanide solutions to an extraordinary degree, there is the possibility of adopting some new method if it can show greater security of gold, ease of clean up, and reduction of space, combined with practically complete extraction of the gold value from very dilute solutions.

It has frequently been a matter of regret to the Rand metallurgists that processes and devices apparently successful elsewhere, have proved unsuited to local conditions. Although fresh information of local or oversea origin and development will always be welcome, the present, "conservative" attitude of preferring actual trial to glowing verbal accounts will continue manifest in Rand metallurgical practice.

NOTE.—For additional discussion of this paper see p. 965.

Slime Agitation and Solution Replacement Methods at the West End Mill, Tonopah, Nev.

BY JAY A. CARPENTER,* E. M., TONOPAH, NEV.

THIS paper deals with only one step in the treatment of ore at the West End mill; not because the other steps are repetitions of practice in other mills, but because in this particular step there is in use the Trent agitator. This device, although it has apparently failed in many mills, is here giving excellent service and has proved to be well adapted for making a thorough replacement of pregnant solution with a minimum amount of barren solution. The strong and weak points of this agitator, its present simple construction, and its use as a thickener for agitator or battery pulp are features of interest; but perhaps the point of greatest interest to the metallurgist in this day of continuous decantation is its use in a series of tanks for slime treatment by the replacement method.

The agitating department of the West End mill consists of six redwood tanks, 24 ft. in diameter, 18 ft. high, each equipped with a Trent agitator, a centrifugal pump, and a motor. The pulp is transferred by a pump from the top of a flat-bottomed tank to a set of arms and nozzles in the bottom of the tank, at just sufficient pressure to cause the arms to revolve. The streams from the many nozzles keep the bottom of the tank clean, and these streams, coupled with the effect of the revolving mechanism and the ascending current, keep the pulp in constant motion and of constant gravity.

These agitating tanks hold 90 tons of dry slime and 202 tons of solution with a 1.24 pulp, our ore having a specific gravity of 2.7. This capacity is about 10 per cent. in excess of the standard 15 by 45 ft. Pachuca tank. The tanks are connected in one series for continuous agitation, the pulp for each agitator being drawn by its pump through a branch suction from near the top of the preceding tank and delivered to the bottom of the tank by the agitator arms along with the regular pulp of that agitator. This method works automatically, since the flow through the branch suction varies directly with the difference in head of the two tanks. The chance for new pulp to pass out quickly is, therefore, much less than in many other systems of continuous agitation, since the new

* Mill Superintendent, West End Consolidated Mining Co.

pulp is delivered at the bottom of the tank, the discharge from the tank is near the top, and the ascending current is uniform and slow.

The pulp is delivered to the first agitator of the series at a gravity of 1.26 by a set of diaphragm pumps, which raise the pulp to a height of 10 ft. above the top of the battery-pulp thickeners, whence it flows first in an open launder and then in a pipe to the suction of the first agitator pump. At intervals in the open launder are placed pieces of battery screen at an angle, which, with those placed at the lip of the diaphragm pumps, catch and remove nearly all the lime scale, wood pulp, and like material from the slow-traveling thickened pulp.

The pulp from the last agitator is raised by an air lift to a Dorr thickener, 28 ft. in diameter and 22 ft. deep, where the pulp is thickened before filtering.

The slaked lime and half the lead acetate are added direct to the scoop box of the tube mills, while the greater portion of the cyanide and the rest of the lead acetate are added at the lip launder of the diaphragm pumps, and the remainder of the cyanide is added in the third agitator. For heating the pulp, live steam is introduced into three of the agitators, Nos. 1, 3, and 5, the purpose being to keep the temperature during agitation around 110° to 120° F., in order to hasten extraction and obtain good extraction with the use of lower-strength cyanide solutions. These high temperatures are readily maintained in a Trent agitator and are not uncomfortable to the operators, since the heavy coat of foam on the agitators acts as an insulating blanket, keeping the pulp from radiating its heat or moisture, as it does in many other types of agitators. The condensed steam adds to the tonnage of mill solution, but it freshens the solution by dilution of impurities and it is needed to replace the solution that is mechanically lost from the filter.

Once a day, at 4 a.m., pulp samples are taken from agitators Nos. 1, 3, 5, and 6, which are thoroughly washed and dried and made ready for the assayer at 7 a.m. Normally, these assays show respectively 50, 25.5, 19, and 15 per cent. of the valuable content of the agitator heads.

Air for the proper aeration of the pulp is compressed to 16 lb. in a compressor of 100 cu. ft. piston displacement, and is fed into the discharge pipe of the agitator pumps. By the time the air is discharged from the nozzles it is thoroughly dispersed in the pulp as minute air bubbles. These air bubbles slowly make their way to the surface and escape into the foam. This well-distributed aeration, in contrast to the short contact of pulp with air in any air-lift type of agitator, is much in favor of the Trent. The pumps of the agitators can be made to suck their own air by throttling the main pulp suction, but in a large installation it is probably more economical to use an air compressor.

The original agitators as installed in 1911 were of the Trent underfeed type, with the large grit-proof bearing in the bottom of the tank and with

the arms fitted with $1\frac{3}{4}$ -in. nozzles. While these agitators gave excellent service, there was a gradual wear, which necessitated repairs on the grit-proof bearing, due to absorption of the air in the air chamber and consequent rise of slime pulp to the ball race. An occasional cleaning of the nozzles was also necessary, due to their choking with bits of wood pulp,

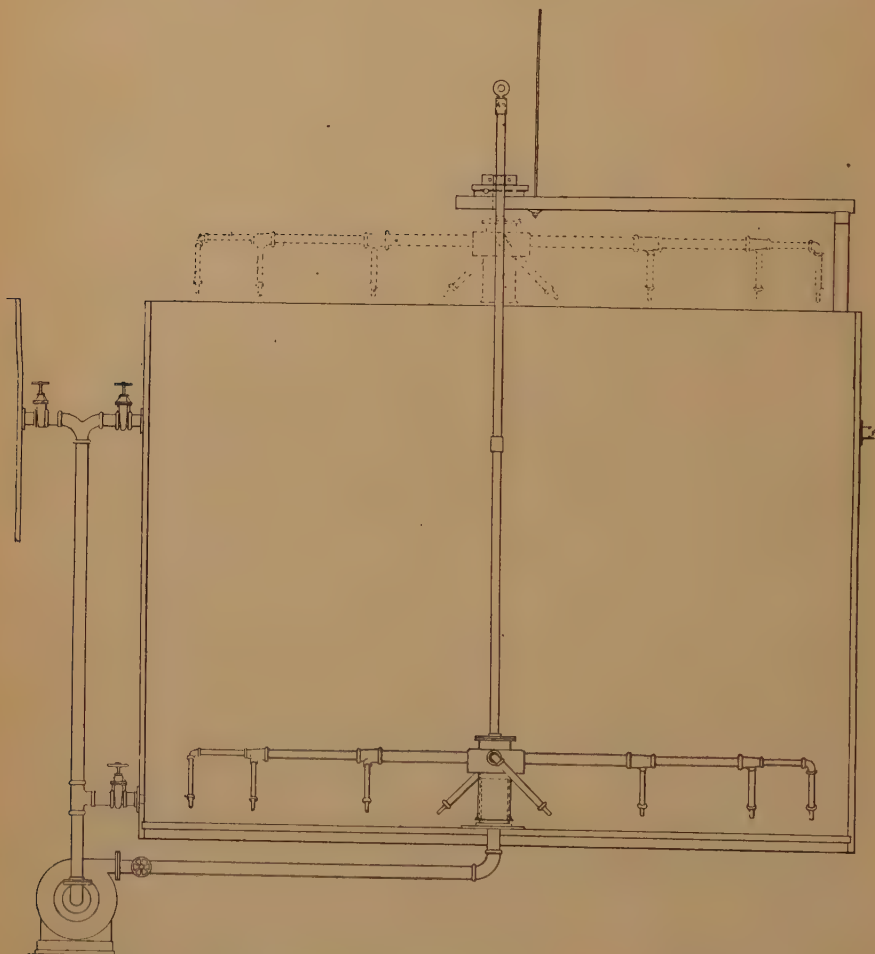


FIG. 1.—TRENT AGITATOR, CARPENTER TYPE.

waste, and chips, as a nucleus, surrounded and packed tight by sand from the pulp. On a charge system these repairs could be taken care of easily when the tank was emptied, but when the continuous system was adopted it was necessary once in two or three months to empty each agitator into the following unit by the aid of the transfer pump, and make these repairs, to avoid the danger of an agitator stopping at a time when

there was not room in the mill circuit to empty it. The author, after seeing a Trent agitator suspended from a ball race at the Tonopah Extension mill, because the lower submerged ball race was worn out, conceived the idea and built a new type of this agitator which could be raised, cleaned, and dropped back into position without disturbing the pulp level in the tank. It consisted of a ball race, on timbers over the center of the agitating tank, from which the agitator mast, built of 3-in. pipe, was suspended, a collar clamped around the upper projection of the pipe resting directly on the ball race (Fig. 1). At the same time, the large upper and lower castings, weighing 800 lb., were discarded, and castings weighing 300 lb. were substituted, which were fitted with extra heavy pipe nipples to serve for the slip joint between the stationary pump discharge and the revolving arms. The upper nipple was of 8-in. pipe, 12 in. long, and was made bell-shaped at the bottom. The lower nipple was of 7-in. pipe, 10 in. long, chamfered at the top. These nipples, after a skim cut in the lathe, have $\frac{1}{64}$ -in. clearance between them and about a 9-in. contact, which allows only a small flow of pulp to pass between them, but permits the agitator to be raised 9 in. before disengaging. An eye bolt was fastened in the roof directly over the center of the tank, and by means of a chain block the agitator is readily raised above the top of the pulp, where every part of the agitator, excepting the 7-in. nipple in the base casting, can be inspected, cleaned, and repaired if necessary. Due to the balanced arms, the bell-shaped nipple, the plumbed point of support, and the guiding ball race, there is no difficulty in lowering the agitator and again engaging the nipples. The first agitator of this type was placed in commission on Oct. 20, 1913. On account of its lighter weight, it traveled faster and put very little weight on the ball race, as the upward reaction from the nozzles nearly counterbalanced the weight. It was raised, cleaned, and lowered with ease. The tank was emptied at first every couple of months to examine the nipple wear, which was found to be very small. The original pair of nipples, after 19 months' service, are still in use and the wear has not affected the speed of the agitator.

After a thorough trial this new type of agitator was placed in all the tanks at convenient times. It has never been necessary to empty an agitator tank because of failure of the agitator of this type to revolve or keep the pulp agitated. The oldest agitator of this type had operated 10 months (on May 15, 1915) and three others nine months without lowering the pulp in the tank; and, judging from the speed of rotation and the feel of the bottom of the tank, they are good for months to come. When the agitator has been in operation for from four to eight weeks a rod drawn along the tank bottom will strike ridges, which are indications of choked nozzles. After raising the agitator and cleaning out an average of a half-dozen choked nipples, the agitator is lowered to within 6 in. of its usual depth until the cleaned nozzles have cut away the ridges.

These agitators have been shut down for hours at a time and have started off after a few minutes' aid in making the first revolution.

This new type of agitator has now been adopted by the Trent Co. as its standard design, adding to the previous good points of this agitator those of simplicity, durability, and reliability.

The pump used on the agitator is a 4-in. centrifugal, driven by a 10-hp. motor. This pump, made by a local foundry, is lined, and has a white-iron open runner. The liners have given an average life of eight months and the runners and shafts of six months; the cost of upkeep of the pump is about \$5 a month. At first, the pumps were driven at 525 r. p. m. and required 6.5 hp. At present, the speed is 460 r. p. m., taking 5.8 hp., and keeping the pulp all of the same specific gravity with the aid of the 1.2 hp. used for compressed air. The speed can be dropped to 360 r. p. m. with good agitation, but not with the constant specific gravity desired for continuous agitation. There is often a false economy in cutting power costs of grinding or agitation a cent a ton, with a probable consequence of twice that loss in extraction. The actual power used, figured as motor input, is 7 hp. per agitator, or about 1 hp. for each 12 tons of ore or 42 tons of pulp. At the Goldfield Consolidated mill 200 tons of ore in a 1.5 to 1 pulp has been agitated for $7\frac{1}{2}$ hp., or 1 hp. for $26\frac{2}{3}$ tons of ore or $66\frac{2}{3}$ tons of pulp.

Several of the earlier failures of the Trent agitator were due to too small an area of nozzle discharge and to improper sizes and speeds of the pumps used. The following tests will illustrate some of the points governing the Trent agitator. With the agitator equipped with $15\frac{3}{4}$ -in. nozzles, with the pump running at 450 r. p. m., and with 3 ft. of pulp in the tank, the pressure on the pump discharge was $7\frac{3}{4}$ lb. With the pump stopped, the pressure from the static head on the discharge was $1\frac{1}{2}$ lb. The actual increase of pressure, due to the pump, was $6\frac{1}{4}$ lb. With the pump running and the discharge valve slowly closed, the pressure increased to 15 lb. and the power decreased one-half. With the agitator run under similar conditions, but equipped with $1\frac{1}{2}$ -in. nozzles, giving four times the discharge area, the pressure on the discharge pipe was $2\frac{1}{2}$ lb., or only 1 lb. increase of pressure above the static head; yet the agitator turned at the same speed of 2 r. p. m. and the motor input of 5.6 hp. remained the same. The reason for the power remaining the same is that a centrifugal pump, running at a given speed, takes approximately the same horsepower for a wide range of heads, but the efficiency, it is to be remembered, varies greatly within this same range. The reason for the agitator running at the same speed is that the greater volume pumped in the second case, acting under less pressure, exerted as much turning force as the lesser volume under greater head.

Figuring from nozzle formula, the quantity discharged in the second case was twice that in the first case, giving, therefore, twice the upward

velocity in the tank. The stronger the upward velocity, the more uniform the specific gravity of the pulp. It is evident, therefore, that a large nozzle area, with a pump designed for handling a large volume at a low head and slow speed, is the best combination, and such a pump, made along the lines of the modern Traylor sand pump, should not need repairs for a period of a year. In both the above cases, as the agitator was filled from 3 ft. in depth to 17 ft., the pressure on the pump discharge increased, but nearly in the same amount as the static head, giving the same difference or actual working head, the same power, and the same agitator speed. Consideration of these points takes the successful installation of this agitator and its pump out of the realm of guesswork.

At times, when the mill heads were high, it became necessary to replace pregnant with barren solution. The battery-tank solution fed to the glands of the pumps acted as an increasing diluent in the continuous circuit, but did not remove pregnant solution. The top suction of the agitator is about 4 ft. below the top of the tank, and by cutting off the air to the agitator this top 4 ft. becomes a static thickener, ending abruptly in the moving pulp at the agitator suction line. The pulp in the tank from this line down will have a constant gravity without the aid of the air. A decanting pipe placed in this top 4 ft. of the agitator removed the pregnant solution direct to the silver tank. The decanting of a full 3-in. pipe stream can begin 30 min. after the air is turned off.

With a crowding of tonnage for several months to over 200 tons a day, which was beyond the capacity of the thickeners ahead of the agitators, the overflow of the four 8-ft. Callow cones in the concentrating department was diverted from the thickeners direct to a well, 2 ft. in diameter and 3 ft. deep, in the center of the above agitator, which was No. 2 in the series. This pulp was very thin and contained the lightest particles of the ore; yet, with all this pulp flowing into the top, this agitator delivered the mill flow of ore in a thicker pulp and of a lower value of solution to No. 3 agitator than it received from No. 1, and, while keeping its pulp in constant agitation for 14 ft. of its depth, it delivered a clear stream of pregnant solution to the silver tank. At another time for several months, with less tonnage but a higher grade of ore, No. 4 agitator was fitted with an overflow launder, and a well, 2 ft. in diameter and 4 ft. deep, to take the feed from No. 3 agitator. The agitator was run without air, with top suction closed, bottom suction open, and a bottom discharge to No. 5 agitator. The pulp entered the agitator at $2\frac{1}{2}$ to 1, and was discharged at 1 to 1 to agitator No. 5, where barren solution was added. The extraction in the tank remained the same as before, when it had air, which, in my opinion, was due to the fact that the greatest extraction had taken place by the time it reached this agitator, and the air included in the pulp was sufficient for extraction. I have known a Dorr thickener, placed in the middle of a series of agitators, to make a greater extraction than the

preceding agitator. However, when the air has been cut off No. 1 Trent agitator in our series, only half the usual extraction has been made. The No. 4 agitator pump on this work was cut to 340 r. p. m., taking 3.5 hp.; yet the agitator started up in its 1.45 pulp after a shutdown as rapidly as the other agitators with only 1.25 pulp. The reason for this is that as the gravity increases the viscosity also increases, hindering the settlement of the heavier sandy particles, which are further handicapped by a diminished difference in gravity between themselves and the surrounding pulp. On this basis, with a proper thickening of the pulp, pyrite concentrate can be readily agitated in a Trent agitator, as is being done at present at the Original Amador Consolidated Mines Co. mill at Amador, Cal.

The next and final step in replacing of solutions was proposed to us by L. C. Trent, the inventor of the agitator. The idea was to feed to the agitator pump a ton of barren solution for each ton of ore fed to the tank, to overflow from the tank the whole tonnage of solution fed to it, and to discharge a 1 to 1 pulp from the bottom of the tank, the solution in which should be the barren solution fed to the pump. This was to be secured by feeding the new pulp to a distributing umbrella on the surface of the pulp in order to feed it in a thin sheet, traveling nearly horizontally, and by having the bottom suction draw pulp from all over the tank by means of a perforated bustle pipe inside the tank, these two doing away with all strong currents. A bypass between the discharge and the suction of the pump would regulate the speed of the agitator, and a series of manifold valves would regulate the flow of barren solution to the pump. A 1 to 1 pulp would be established in the bottom 6 ft. of the tank, and above this a sharp change to thin pulp, and then to clear solution at the top of the tank. The new pulp entering the tank would enter on the surface, only causing a ripple. The solid content would settle through the clear zone into the thick pulp below, but the solution would overflow into the launder, since there would be no downward current, due to the fact that as much solution was introduced at the bottom of the tank as was removed. For a similar reason, the solution fed in at the bottom would find no rising current, and, being mixed with the pulp, would be discharged as an integral part of the discharge pulp. By this method, the inventor reasoned that it would easily be possible to make a 90 per cent. replacement of the dissolved silver entering the tank.

At the time of this visit, every agitator was needed for agitation purposes, the canvas-leaf filter was doing good work, and, lastly, we had little faith in making a high replacement of pregnant solution by this method, for all the many efforts along this line had been failures; and it was generally conceded that the diffusion of solutions of equal gravity took place so quickly and so completely that such a scheme was not practical. However, these replacers were installed at the Gold Cross and

Imperial Reduction Co. plants near Ogilby, Cal., and news of their successful operation came to us not only from the company headquarters, but from the traveling fraternity of millmen looking for a job.

In February, 1915, with a decrease in tonnage, No. 4 agitator was fitted up for a trial as a replacer. With the agitator filled with 1.20 specific gravity pulp, containing 203 tons of solution assaying 7 oz. silver, the feed of pulp and the air were cut off, and the bottom suction opened. Three hours later, when the pulp in the bottom of the tank reached 1.36 specific gravity, a flow of barren solution, assaying 0.13 oz. silver per ton, was admitted to the pump at the rate of about 3 tons an hour. At the end of 12 hr. the gravity at 4 ft. from the bottom was 1.36, at 8 ft., 1.11, and at 12 ft., clear; and the solution assays were 3.94, 4.57, and 6.97 oz., respectively. In 36 hr. there had been added a little over 100 tons of solution, and the specific gravity at 4 ft. from the bottom was 1.34; at 8 ft., cloudy; at 12 ft., cloudy, and the overflow clear; and the solution assays were 1.79, 2.52, 6.21, and 6.6 oz., respectively. Since there was no discharge from the tank, the addition of barren solution had to be slow to keep from thinning the pulp in the bottom of the tank. The replacing action is distinctly shown by the assays. At the end of this time pulp feed and pulp discharge were started into and from the replacer. The results of the next two days' work are shown in Table I.

The sharp reduction in the value of the ore, due to the extraction made in the replacer, makes it difficult to calculate the efficiency of the replacement, but, since most of the ore is in the bottom 6 ft. of the tank in the barren-solution zone, it is probably true that most of the dissolved silver goes out with the discharged solution.

During the second day there was more solution discharged from the tank than barren solution fed into the pump, and the effect is shown by the assays.

On the third day the agitator slowed down, and shutting the bypass did not increase the speed; yet the turning pressure of a man's hands on the ball race would turn the agitator faster. It pointed clearly to a choked suction pipe, which, with an inventor's optimism, was submerged under 12 ft. of pulp. Not being able to continue the experiment, the top suction was opened and the agitator started off at 3 r.p.m. The result of the experiment demonstrated that a barren zone could be maintained in the bottom of the tank and a high percentage of replacement made, but it also showed that the capacity of the 24-ft. tank was limited to 60 tons of ore per day at a $2\frac{1}{2}$ to 1 dilution, in order to keep a clear overflow, and proved that the mechanical arrangement was defective.

To improve the mechanical defects, the author designed both a new arrangement for the incoming pulp and a new suction for the replacer pump. Instead of the umbrella, a well, 2 ft. in diameter and 4 ft. deep, was placed at the center of the tank around the mast. The bottom was

made solid and the sides perforated with $\frac{3}{4}$ -in. holes. An arm was put on the mast, which kept the bottom of the well clear.

The suction of the pump was carried around the outside of the tank as a bustle pipe, with the diameter proportioned to the flow. Twelve $1\frac{1}{2}$ -in. branches entered the tank at the floor level and ended at varying lengths in upturned ells. Each branch suction was connected to the main pipe by a plug cock, and each had a $\frac{3}{4}$ -in. connection for testing the flow and for connecting high-pressure water if it was choked.

On Mar. 5, 1915, the replacer was started again with these changes. The new well still introduced the incoming pulp horizontally, but below the surface, giving a greater distance down to the slime line at the overflow launder. The new suction made it possible to test and keep open the 12 suction ports at all times, and left nothing inside the tank to give trouble. The agitator, being the new style, was capable of being raised out of the tank for cleaning the nozzles, and for repairs if necessary, and dropped back without affecting the solution zones in the tank. In starting the experiment, the introduction of the barren solution, as before, gradually established a barren zone, or, rather, in this case, a zone of low

TABLE I

Time Date Shift		Pulp Feed							Overflow Solution			13 Ft. Valve		
		Spec. Grav.	Ratio Solution to Ore	Ore			Solution			Tons	Oz. per Ton	Total Oz.	Spec. Grav.	Oz. per Ton
				Tons	Oz. per Ton	Total Oz.	Tons	Oz. per Ton	Total Oz.					
First Test:														
1/28	3-11	1.24	2.25	22.7	4.90	122	51.2	7.91	405	70.0	6.66	466	1.05	5.66
	11-7	1.23	2.37	21.9	4.56	100	51.8	7.41	384	61.0	6.60	402	1.05	5.63
	7-3	1.21	2.63	24.2	4.14	100	63.8	7.02	448	50.0	6.47	323	1.05	5.82
24 hr.		68.8	4.68	322	166.8	7.42	1,237	181.0	6.58	1,191
1/29	3-11	1.19	2.95	18.1	4.02	72	53.1	6.82	362	74.0	6.36	471	1.06	6.29
	11-7	1.21	2.63	18.6	4.49	83	49.1	6.88	338	34.0	6.35	216	1.06	6.50
	7-3	1.20	2.78	21.5	4.41	95	59.9	6.82	408	50.0	6.83	341	1.06	6.59
24 hr.		58.2	4.31	250	162.1	6.83	1,108	158.0	6.50	1,028
Second Test:														
3/7	24 hr.	1.25	2.15	63.6	2.23	142	108.0	5.62	607	120.0	4.40	530	1.07	4.59
	7-3	1.25	2.15	22.7	49.0	6.84	50.0	1.09
	3-11	1.22	2.50	21.5	54.0	6.84	51.0	1.09
	11-7	1.23	2.37	18.6	44.0	6.84	45.0	1.11
3/9	24 hr.	1.23	2.34	62.8	2.32	146	147.0	6.84	1,005	146.0	5.20	760	5.24
	7-3	1.23	2.37	19.0	45.0	1.09
	3-11	1.21	2.63	17.9	42.0	1.10
	11-7	1.23	2.37	16.5	43.0	1.10
3/10	24 hr.	53.4	2.32	124	130.0	6.80	882	155.0	5.60	866	5.72
	7-3	1.20	2.78	16.8	45.0
	3-11	1.20	2.78	19.1	53.0
	11-7	1.19	2.95	20.6	61.0	1.09
3/11	24 hr.	56.5	2.24	126	159.0	5.66	900	184.0	5.30	970	5.28
	7-3	1.18	3.13	18.0	2.24	40	56.0	5.62	314	55.0	5.39	296	5.30
Totals:														
First Test.....		127.0	4.50	572	328.9	7.13	2,345	339.0	6.54	2,219
Second Test.....		254.3	2.30	585	600.0	6.18	3,708	660.0	5.17	3,416

grade at the bottom, increasing in value upward in the tank. This suggests a method for very small plants, such as used on old dumps, of using the replacer as a thickener first, an agitator second, and lastly as a replacer, discharging after making a second replacement with water. After establishing the barren zone, the feed into and discharge from the replacer were started, and the results obtained are shown in Table I.

A detailed study of the table brings out many interesting features of the replacer. A feed of replacing solution considerably in excess of discharge solution for the period of a shift results in a marked lowering of the solution value in the thick pulp and a decrease in that of the thin pulp, while the opposite condition of feed will increase the value likewise. The sharp change in the silver content of the solution is between the thick 1 to 1 pulp and the thin pulp above it, for the reason that the diffusion in the thin pulp is rapid. Hence, there is no advantage, in installing a replacer, in having any more depth above the 6 ft. of thick pulp than is necessary for the settling of the ore. It is possible to discharge the pulp thicker than 1 to 1, but it takes more power applied to the pump to turn the agitator. With a positive drive overhead, taking

TABLE I—Continued.

9 Ft. Valve		4 Ft. Valve		Pulp Discharge									Replacing Solution		Solution	
				Spec. Grav.	Ratio Solution to Ore	Ore			Solution							
						Tons	Oz. per Ton	Total Oz.	Tons	Oz. per Ton	Total Oz.					
Spec. Grav.	Oz. per Ton	Spec. Grav.	Oz. per Ton													
1.29	4.99	1.41	2.41	1.38	1.29	5.1	4.13	21	6.6	2.10	14	14.0	0.20	3.0		
1.23	4.04	1.43	2.34	1.45	1.03	10.2	4.17	42	10.4	2.22	23	18.7	0.20	3.7		
1.07	4.88	1.42	2.40	1.42	1.13	26.5	4.01	106	30.0	2.17	65	20.2	0.23	4.6		
.....	41.8	4.07	169	47.0	2.19	102	52.9	0.21	11.3		
1.06	6.04	1.45	3.00	1.43	1.10	21.4	4.02	87	23.6	2.60	61	16.3	0.27	4.4		
1.07	6.25	1.49	3.20	1.46	1.00	22.9	3.93	90	22.9	2.73	62	15.3	0.29	4.4		
1.09	6.25	1.45	3.10	1.42	1.13	29.5	3.91	114	32.9	2.60	85	25.0	0.29	7.5		
.....	73.8	3.95	291	79.4	2.62	208	56.6	16.3		
1.17	4.44	1.47	1.83	1.46	1.00	43.4	1.70	74	43.4	1.52	66	50.4	0.16	8.0		
1.12	1.45	1.43	1.08	22.8	24.4	1.30	32	25.5		
1.10	1.46	1.46	1.00	24.0	24.0	1.52	36	21.3		
1.11	1.48	1.48	0.94	20.4	19.2	1.58	30	20.2		
.....	4.60	1.91	67.2	1.92	129	67.2	1.46	98	67.0	0.16	10.7		
1.10	1.45	1.46	1.00	19.5	19.5	1.63	32	23.9		
1.12	1.48	1.48	0.94	13.5	12.5	1.72	21	24.0		
1.10	1.48	1.49	0.92	13.3	12.2	1.70	21	22.6		
.....	5.63	2.36	46.3	1.92	89	44.2	1.68	74	70.5	0.15	10.5		
.....	1.48	0.94	13.0	12.2	1.53	19	19.0		
.....	1.47	0.95	20.2	19.0	1.45	27	24.6		
1.10	1.52	1.51	0.87	17.3	15.0	1.67	25	22.0		
.....	5.02	1.80	50.5	1.90	96	46.2	71	65.6	0.16	10.5		
.....	5.00	1.90	1.50	0.89	20.9	1.90	39	18.7	1.37	26	22.0	0.17	3.7		
.....	115.6	3.98	460	126.4	2.46	310	109.5	0.25	27.4		
.....	228.3	1.87	427	219.7	1.52	335	275.5	0.16	43.4		

very little power, the pulp could be discharged as an 0.8 to 1 pulp or thicker with better replacing results.

In order to calculate the replacement of solutions in these tests, two methods are used as checks: First, dividing the ounces overflowed by the ounces fed; second, dividing the ounces discharged by ounces fed. However, before doing this, the ounces extracted from the ore and the ounces fed in with the replacing solution should be divided between the ounces overflowed and ounces discharged. These two items amount to considerable in our tests, because we were not equipped to add barren solution, and, secondly, the ore still lacked sufficient agitation to be ready for filtering. Since the replacing solution is fed to the bottom of the tank, and assays show that it remains there as a partly barren zone, and since over 90 per cent. of the ore is in the bottom half of the tank and a large part of it in contact with the barren replacing solution, it is fair to presume that over 80 per cent. of the replacing solution and ore extraction go out in the discharge and 20 per cent. reaches the overflow.

In the first test, 27 oz. of silver was introduced with the replacing solution, and 112 oz. of silver was yielded to the solutions by the ore, or a total of 139 oz., 20 per cent. of which is 28 oz. and 80 per cent. 111 oz. Silver to the amount of 2,345 oz. was fed with the pulp solution, and of this 2,219 oz. less 28 oz. was overflowed, giving 93.2 per cent. replacement. Figuring by the second method, of the 2,345 oz. of silver fed, only 310 oz. less 111 oz. of it was discharged, giving 91.5 per cent. replacement.

In the second test 43 oz. of silver was introduced with the replacing solutions and 158 oz. of silver derived from the ore. The silver fed with the pulp solution was 3,708 oz. and of this 3,416 oz. less 40 oz. was overflowed, giving 93.2 per cent. replacement. Figuring by the second method, of the 3,708 oz. fed, only 335 oz. less 160 oz. of it was discharged, giving 95.3 per cent. replacement.

In actual practice there is always a little silver in the replacing solution, and, as has been found in the continuous-decantation method, there is always an extraction from the ore, no matter how long it has been treated before decantation. These items were considered in the foregoing calculations, as they were exceptionally high under our conditions. Again, in the tests, 11 per cent. more replacing solution was added than was discharged, which could be done with the final water replacing, due to the building up of solution tonnage.

However, it was established to our satisfaction:

1. That replacing with the Trent apparatus is practicable.
2. That it is easy to control and regulate the replacer; the one important factor being to regulate the solution fed to the pump and that discharged from the replacer. In a plant designed for this system this would be a much easier matter than in our case.
3. That on a quick-settling pulp it would handle a remarkably large

tonnage as far as adding the necessary barren solution is concerned, but that on a pulp carrying a fair percentage of light, flocculent slime it is limited in tonnage to the rate at which the light, flocculent slime will settle in a neutral or no-current tank. In a Dorr thickener there is a downward current to aid settling. At the new mill at Aurora, Nev., the light slime hardly settled in a Dorr thickener, and practically remained in suspension in the Trent replacer.

4. That mechanically the replacer worked very satisfactorily, but that changes could be made to cut down the power consumed and yet make the rotating action more positive.

5. That with a mixture of Tonopah ores, 60 tons of ore in a $2\frac{1}{2}$ to 1 pulp, with 60 tons of replacing solution, was all that a replacer 24 ft. in diameter and 18 ft. high could handle with a clear overflow and a 1 to 1 discharge.

The use of this agitator for replacer experiments necessitated bypassing all but 60 tons past this agitator, and caused a loss in extraction that made it necessary to stop the experiment without a working test extending over a few weeks' time; but the author believes the same results would have continued, as everything about the installation was capable of easy adjustment and repair.

Before changing back to an agitator, the replacer was run as an ordinary thickener with a 1 to 1 discharge, all replacing solution and gland water being cut off the pump, and the speed of the agitator cut to one revolution in 5 min. Hose lines were strung to add a large tonnage of solution to the 60 tons of ore in a $2\frac{1}{2}$ to 1 pulp. To run as a thickener and make 90 per cent. replacement with a 1 to 1 discharge, the 60 tons of ore would have to be diluted to a 10 to 1 pulp by the addition of 450 tons of barren solution to the 150 tons of pregnant solution. About a 7 to 1 pulp was all that could be supplied without interfering with mill operations, but the slime line rose rapidly from 4 ft. below the surface up to the overflow launder. The 60 tons capacity of the replacer had appeared disappointing, but the test showed it to be a good tonnage on this particular ore compared with replacement by dilution; but it also confirmed the author's conviction that on an ore containing considerable flocculent slime the various replacing methods will have a strong rival in the older leaf and pressure filters that make an easily washed cake from the mixture of the flocculent and sandy components of the ore.

The great advantage of the replacer over the thickener-dilution method is in the smaller installation required, and the small amount of barren solution required compared with the large tonnage of barren and partly barren solution of the dilution method.

The main drawback to the replacer system has been, first, lack of faith in its principles, and second, lack of faith in its mechanical features.

To one who has operated the replacer it is easy to understand that in a

1 to 1 and thicker pulp, the pulp is so thick that the ore and solution are approaching a solid, and that the viscosity of the pulp is so great that the solution is not free to move about amid the ore particles, but, rather, is constrained to move with them; and that, once the barren solution is whipped into the pulp by the pump, it tends to remain locked up in it. The pump is of such a size that the difference in specific gravity between its suction and its discharge is only a couple of hundredths, so that the discharge pulp has no tendency to rise in the tank. This explains why the Trent method is a success and why other methods of adding the replacing solution directly into the bottom of a tank have failed.

The last year has brought about a marked improvement in mechanical simplicity and reliability in the mechanism of the replacer. One strong objection to it, compared with the dilution system, is the power required to operate it. However, this does not seem excessive when one considers the saving of the power that is consumed in handling and precipitating the large tonnages of solution required by the dilution methods. The author believes the power consumption of the replacer can be made nearly as low as that required by the equivalent number of thickeners, by the simple device of not depending entirely upon the pump to rotate the agitator, but to drive the agitator from overhead by a worm drive similar to that of the Dorr thickener and to use the pump more as a mixer and circulator of pulp, running it at a slower speed and with larger nozzles on the agitator. Our experience was that it required about 3 hp. on the motor to turn the agitator at only 1 r. p. m. in a 1 to 1 pulp, when a man with a 1-ft. leverage on the mast could turn it with ease. Why, then, should pump nozzle velocity be relied upon to turn the agitator when its proper function should be that of mixing the barren solution into the pulp and giving only the turning effect derived from that function? The pump, with a slower speed and lower working head, should need only annual repairs.

With these modifications, the author believes that the replacer will become an accepted machine in the treatment of slime that is not too colloidal in its nature.

The Tonopah Plant of the Belmont Milling Co.

BY A. H. JONES,* TONOPAH, NEV.

(San Francisco Meeting, September, 1915)

THE Belmont mill at Tonopah, Nev., was designed and constructed by the Belmont staff. Ground was broken in August, 1911, and milling operation started July 25, 1912. The metallurgical flow sheet, and the machinery adapted to carrying out operations in accordance with this plan, were decided upon from experience gained in the treatment of Belmont mine ores at the old Belmont mill at Millers, Nev.

The detail drawings were made by the engineering staff of the Tonopah-Belmont Development Co., under the supervision of Otto Wartenweiler, who also had direct charge of the construction, consulting with the General Superintendent of the company, Frederick Bradshaw, and the writer.

TABLE I.—*Construction Costs, Belmont Mill*

	Excavation Concrete Walls and Founda- tions	Floors and Machinery Founda- tions	Buildings		Machinery, Including Erection, Piping, Wiring, Belting, Etc.	Totals
			Frames	Covering		
Crusher plant.....	\$5,760.72	\$2,527.58	\$2,230.72	\$1,476.69	\$21,174.96	\$33,170.67
Inclined conveyor.....	166.41	238.02	1,771.75	585.19	3,620.79	6,382.16
Battery bins.....	399.00	489.70		2,067.69	3,475.14	6,431.53
Stamps.....		8,797.82			36,873.06	45,670.88
Tube mills and classifiers...		3,180.95			40,127.89	43,308.84
Callow cones.....		26.02			856.63	882.65
Concentrating plant.....		1,663.23			11,356.35	13,019.58
Concentrate house.....	76.80	449.69	354.80	623.27	602.10	2,106.66
Dorr thickeners.....		11,297.98			15,034.53	26,332.51
Circulating system.....		147.81			6,846.05	6,993.86
Air agitation.....	261.71	3,987.80			25,257.89	29,507.40
Clarifying.....		1,829.63			8,573.87	10,403.50
Precipitation system.....	395.00	44.05			29,581.02	30,020.07
Briquetting plant.....					1,589.49	1,589.49
Air compressor.....		1,084.39			7,094.52	8,178.91
Filter plant.....		4,456.68			30,104.50	34,561.18
Refinery.....	2,473.84	1,552.23	2,200.17	2,292.80	7,548.79	16,067.83
Boiler plant and fuel-oil system.....	571.56	90.97	401.25	531.11	7,606.28	9,201.17
Tank-heating system.....		9.56			3,362.39	3,371.95
Transformer house.....	91.83	101.05	428.69	213.07	4,804.89	5,639.53
Lime house.....	11.00			753.30	305.85	1,070.15
Machine shop.....	1,297.66	843.01	1,138.73	1,191.93	339.36	4,810.69
Storeroom.....	511.55	1,509.60	1,305.84	1,315.78	152.04	4,794.81
Inclined railway.....	133.25		432.70	203.41	379.51	1,148.87
Mill building.....	39,645.45	6,757.60	45,493.48	19,607.14	9,020.51	120,524.18
Total.....	\$51,795.78	\$51,085.37	\$55,758.13	\$30,861.38	\$275,688.41	\$465,189.07

* Superintendent of Mills, Tonopah-Belmont Development Co.

The total cost of the plant, designed to handle 500 tons per day, was \$465,189.07. The saving in operating costs and increased extraction, as compared with the best results from the old mill at Millers, was such as to return the entire cost of the new plant from the treatment of 260,000 tons of ore, or approximately 18 months' operation. Fig. 1 is a view of the plant, and Fig. 2 a section of the mill.

Segregated construction costs are given in Table I.

In order to present the matter clearly, a general description of each step of the milling operation will first be given and the metallurgical process considered, after which the operation will be gone over again, step by step, and discussed from a mechanical and economic standpoint. The general flow sheet is given in Fig. 3.

GENERAL DESCRIPTION AND METALLURGY

Crushing and Conveying

All crude ore is broken underground to pass a 9-in. grizzly and hoisted in 3-ton skips to the ore pockets, 37 ft. above the collar of the shaft. From the ore pockets it is conveyed in a 2-ton car by a Hunt automatic railroad to the crusher pockets.

Two circular steel crusher pockets, with a combined capacity of 1,000 tons—about 500 tons of which will run freely—are situated one on each side of the picking belt. Rock is fed from the crusher pockets through finger gates, hand operated, over shaking grizzlies with 2-in. openings. The undersize bypasses the picking belt, going direct to the trommel, and the oversize is fed to a steel picking belt 40 in. wide, with 50-ft. centers, traveling 45 ft. per minute, from which belt seven ore sorters pick the waste and throw it down chutes on to a 20-in. conveyor belt which discharges upon the waste dump.

The picking belt discharges over the shaking feeder to a No. 7½K gyratory crusher set at 2-in. ring. The discharge from this crusher joins the undersize from the crusher pockets to form the total feed for the trommel, which is 48 in. in diameter by 14 ft. long, with 1¼-in. openings.

The undersize from the trommel goes direct to the inclined conveyor belt, and the oversize forms the feed for two No. 4D short-head gyratory crushers set to 1-in. ring.

The discharge from the No. 4 crushers and the undersize from the trommel are delivered to a 20-in. conveyor belt, with 250-ft. centers, on an incline of 19° 20', which carries the ore to the head of the mill.

While the ore is passing over this belt it is automatically weighed by an electric weighing machine.

From the head of the incline the ore is conveyed and distributed, by means of a horizontal belt conveyor and automatic tripper, to flat-bottom



FIG. 1.—VIEW OF THE PLANT OF THE BELMONT MILLING CO.

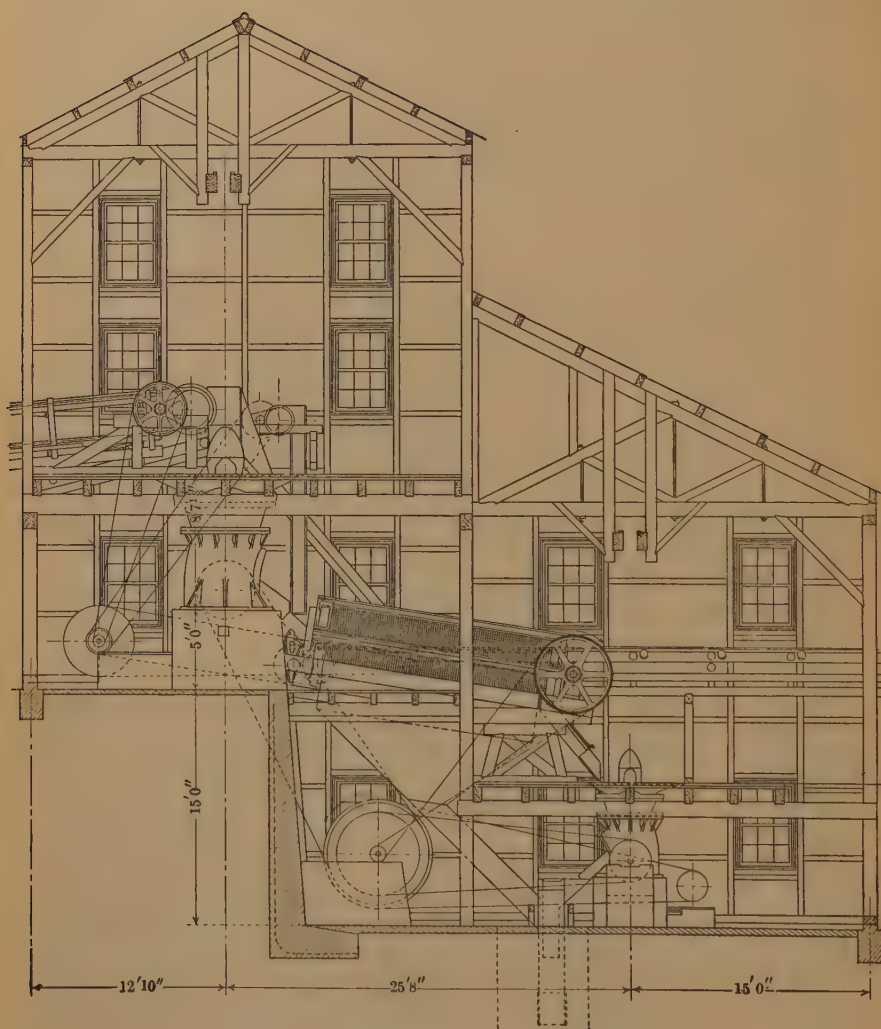


FIG. 2.—SECTION OF BELMONT MILL.

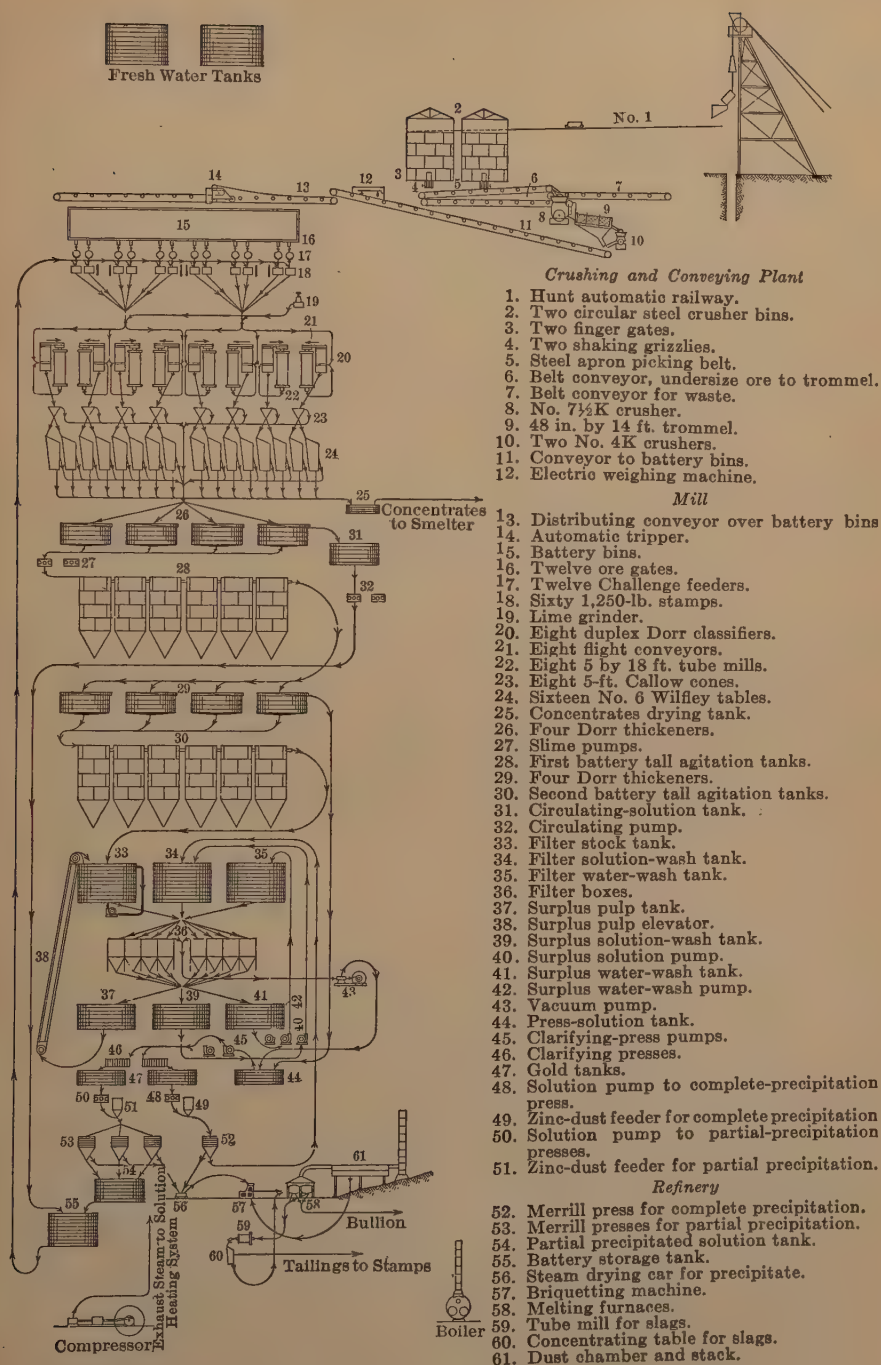


FIG. 3.—FLOW SHEET OF BELMONT MILL.

battery bins, 16 by 17 by 110 ft. in the clear, with a capacity of 1,500 tons, about one-half of which will run freely.

Sampling

At the point where the inclined belt discharges on to the horizontal belt is placed a device consisting of a 74-tooth gear with 24 teeth removed and a 25-tooth pinion, with a bucket of about $5\frac{1}{2}$ lb. capacity attached to the gear. The gear makes one revolution every 68 sec., traveling very slowly until the bucket reaches the stream of ore, when a counterbalance actuates the gear through the part with the teeth removed, allowing the bucket to make a quick cut of the ore. The sample thus taken is further crushed in a McCully laboratory gyratory crusher to pass $\frac{1}{4}$ -in. ring and riffled down to about 50 lb., which is further reduced and cut at the assay office for a representative sample of the ore crushed during the day. While this original cut, of about 1 ton in 500, seems small, it was checked for 3,500 tons against the sampling of the Western Ore Purchasing Co. and averaged within 8c. per ton of their results.

As an accurate sampling with this grade of ore cannot be had without a very elaborate sampling plant, and at a considerable operating cost per ton, while the tailings or discharge from the mill is a product easily sampled, being thoroughly mixed and of such low value that a large error is not possible, the actual value of the ore treated for any month is figured from bullion produced, plus concentrates made, plus tailings discharged. The daily sampling of ore milled is entered in the mill books and used simply as a check against the other method of figuring the metallurgical results.

Stamping

From the battery bins the ore is fed through open gates and Challenge feeders to sixty 1,250-lb. stamps, which crush through 4- and 6-mesh Tyler "ton-cap" screens. For the fiscal year Mar. 1, 1914, to Feb. 28, 1915, the average stamp duty was 8.88 tons per day.

Stamping is done in cyanide solution, about 5 parts of solution to 1 of ore being introduced into the mortar, the solution having a cyanide strength of 5 lb. and a lime strength of about 1 lb. CaO. The discharge is kept about 3 in. above the dies. A screen analysis, with one-half 4-mesh and one-half 6-mesh screens, is given on the following page.

The launders for the 30 stamps on each side are so arranged, the discharge lip of each battery of five stamps being divided at the center, that any $2\frac{1}{2}$ stamps can deliver to any one of the four tube mills on that side of the mill. This is accomplished without an elaborate or cumbersome launder system by carrying a four-compartment launder along the retaining wall below the stamps, each compartment delivering to its respective

tube mill. From each $2\frac{1}{2}$ stamps there is a short, closed-end launder, with a hose discharge that can be changed to any compartment of the main launder system.

Screen Analysis of Battery Discharge

Mesh	Per Cent.
+ 10.....	10.0
+ 20.....	19.0
+ 40.....	22.0
+ 80.....	15.0
+100.....	3.5
+150.....	4.0
+200.....	2.5
-200.....	24.0
	<hr/> 100.0

Tube Milling

The battery discharge is delivered to eight Dorr duplex classifiers, placed parallel with their respective tube mills, and at such an elevation that a spiral feeder with a 36-in. lift will deliver coarse feed to the mill and at the same time allow the tube discharge, together with the battery discharge, to flow, with no other elevation, into the classifier and form a closed circuit.

Sufficient lime for one 8-hr. shift is slaked in a grinding pan, used as a mixer, and the overflowing milk of lime is introduced with the tube-mill feed. At this point about 0.15 lb. of lead acetate—approximately one-half the total amount used in the mill—is added.

For some time after the mill commenced operation, the grinding was kept well above 80 per cent. through 200 mesh, with eight mills, but experimental work has since demonstrated that the higher extraction obtained with 85 per cent. through 200 mesh, over 75 per cent. through 200 mesh, would not compensate for the cost of the further reduction, and we are at this time finishing about 75 per cent. through 200 mesh, with seven mills.

There has been a great deal of controversy as to the amount of return feed when running tube mills in closed circuit, and to check this return a central baffle was put into a Dorr duplex classifier, giving the initial feed to one side and the return to the other side, and the tonnages from both sides carefully calculated. This test gave the following figures: Total feed 127.38 tons, initial feed 67.18 tons, return feed 60.20 tons. Ratio of return to initial feed 0.91 to 1. Screen analyses and tonnages are given in Table II.

TABLE II.—Tonnes and Screen Analyses for Classifier and Tube Mill in Closed Circuit

Mesh	Battery Discharge Tonnage by Weighed Sample			Battery Tube Feed Tons by Weighed Sample			Battery Dorr Overflow Tonnage by Difference between Battery Discharge and Battery Tube Feed			Tube-Mill Discharge Tonnage by Battery Feed and Return Feed			Return Feed Weighed Feed			Return Dorr Overflow Tonnage Battery Feed to Tube			Dorr Overflow General Battery Discharge Tonnage		
	Per Cent.		Tons	Per Cent.		Tons	Per Cent.		Tons	Per Cent.		Tons	Per Cent.		Tons	Per Cent.		Tons	Per Cent.		Tons
	Cumulative	Per		Cumulative	Per		Cumulative	Per		Cumulative	Per		Cumulative	Per		Cumulative	Per		Cumulative	Per	
+ 10	8.2	8.2	6.02	8.2	8.2	5.51
+ 14	12.3	20.5	8.03	13.1	21.3	8.80
+ 20	11.8	32.3	8.66	12.2	33.5	8.19
+ 28	12.2	44.5	8.95	13.1	46.6	8.80
+ 35	18.9	53.4	6.53	9.8	56.4	6.58
+ 45	7.9	61.3	5.79	9.1	65.5	6.12
+ 65	5.1	66.4	3.74	7.0	72.5	4.70	Trace
+ 100	5.0	71.4	3.67	10.5	82.5	6.72	2.5	0.155	21.625	53.625	27.55	34.75	55.83	20.90	1.375	0.92	1.375	1.375
+ 150	4.0	75.8	3.23	10.5	93.0	7.05	9.125	11.625	0.565	26.25	59.875	33.42	30.58	36.43	18.40	13.75	15.125	9.24	11.25	12.625	8.26
+ 200	3.2	78.0	1.61	2.2	95.2	1.48	6.0	17.625	0.372	8.5	68.375	10.83	3.45	91.83	3.26	10.625	25.75	7.14	7.875	20.5	5.78
- 200	22.0	100.0	16.15	4.8	100.0	3.23	82.375	100.000	5.108	31.625	100.000	40.29	8.17	100.00	4.92	74.25	100.00	49.9	79.5	100.0	58.35
Total Tons	73.37	67.18	127.38	60.21	67.20	73.40

A simple and accurate method of figuring this return tonnage, in accordance with the screen table below, is as follows:

Mesh	Battery Discharge		Tube-Mill Feed		Tube-Mill Discharge		Dorr Overflow	
	Per Cent.	Cumulative, Per Cent.	Per Cent.	Cumulative, Per Cent.	Per Cent.	Cumulative, Per Cent.	Per Cent.	Cumulative, Per Cent.
+ 10	10.3	10.3	5.4	5.4
+ 14	10.0	20.3	5.2	10.6
+ 20	10.2	30.5	5.3	15.9
+ 28	11.3	41.8	5.9	21.8
+ 35	7.7	49.5	4.9	26.7	1.0	1.0
+ 48	7.1	56.6	6.5	33.2	3.1	4.1
+ 65	5.6	62.2	8.1	41.3	6.5	10.6
+100	6.0	68.2	20.0	61.3	20.0	30.6	1.0	1.0
+150	6.6	74.8	24.4	85.7	28.6	59.2	12.5	13.5
+200	4.0	78.8	7.8	93.5	12.7	71.9	14.2	27.7
-200	21.2	100.0	6.5	100.0	28.1	100.0	72.3	100.0

Tons crushed per day, 500. Feed per mill, 62.5 tons.

In the battery discharge there was 41.8 per cent., or 26.12 tons, which did not show in the tube-mill discharge. This material was represented in total tube-mill feed by 21.8 per cent. As we know this 21.8 per cent. is represented by 26.12 tons, 100 per cent. of the feed amounts to 119.8 tons.

Total feed, tons.....	119.8
Initial feed, tons.....	62.5
Return feed, tons.....	57.3
Ratio of return to initial feed.....	0.92 to 1

Concentration

The overflow from the Dorr classifiers, or the finished product from the tube mills, flows to eight 5-ft. Callow cones, used as sloughing-off cones, the overflow going from there direct to the Dorr thickeners, and the spigot discharge forming the feed for 16 No. 6 Wilfley concentrating tables, running 233 strokes per minute with a $\frac{5}{8}$ -in. throw.

Close concentration is not attempted, the intention being to take out only the heaviest and most refractory material. This method gives a very clean concentrate product, but not a close concentration.

The concentration for the fiscal year Mar. 1, 1914, to Feb. 28, 1915, was 0.67 per cent. by weight and 11.5 per cent. in extraction of precious metals.

The concentrates are trammed from boot boxes to a vacuum tank, 14 ft. in diameter by 3 ft. deep, so arranged that it forms the bed of a 15-

ton card-recording scale, and built over a concentrate storage bin. In this tank the concentrates are allowed to dry by vacuum for 48 hr. when they are weighed and shoveled through a center gate to the steel bin below and sampled for both moisture and metallic content. The elevation of the floor of this bin is the same as that of a railroad freight car. At the time of shipment the entire lot is sampled by saving every fifth shovel, this sample being cut by saving each third shovel, leaving about 3 tons, which is quartered down to final sample, the reject at all times being wheeled direct to the freight car. By this method sampling and loading are finished at the same time.

A great deal of experimental work has been done with a view to treating concentrates on the ground, and while a very attractive extraction can be obtained with new solution, without regeneration the solution would soon become so foul and inactive that a satisfactory extraction could not be made. And even under the most favorable conditions the cost of treatment would be slightly higher than the shipping and marketing expense. It should be stated in this connection that after roasting concentrates a satisfactory extraction cannot be obtained with cyanide.

An analysis of concentrates is given below:

	Per Cent.
S.....	31.60
Insoluble.....	30.60
Fe.....	29.80
Pb.....	1.30
Zn.....	0.60
CaO.....	0.80
Mn.....	1.10
Cu.....	0.60
Au and Ag.....	1.60
Al ₂ O ₃	0.42
Undetermined.....	1.58
Total.....	100.00

First Thickening

The pulp from the Wilfley tables and the overflow from the Callow cones flow to four 30 by 12 ft. Dorr thickeners. The ratio of solution to ore is about $8\frac{1}{2}$ to 1, added as follows: 5 or 6 parts at the batteries, about 0.8 for correction of moisture at the tube-mill feed, about 1 part at the Dorr classifiers in order to thin the pulp for proper classification, and about 1 part as wash water on the concentration tables.

The four 30 by 12 ft. Dorr thickeners allowed 5.5 sq. ft. of settling area per ton of ore, and this was adequate for the first year of operation, but since then development in the mine of veins narrower than the main Belmont vein—and from which a large tonnage has been produced—has

tended to send to the mill a material much more talcose than formerly, and to overcome this difficulty Dorr settling trays were installed in two of the four primary thickeners, allowing, on a 500-ton per day basis, 8.47 sq. ft. of settling area per ton of ore every 24 hr.

These settling trays have worked out to very good advantage, giving no serious trouble in a mechanical way and increasing the capacity of the two 30 by 12 ft. thickeners in which they are installed by about 75 per cent.

Agitation

The overflow from the lower or first thickeners flows either to the circulating tank or to the precipitation supply tanks.

The underflow, at about 1.26 specific gravity, is pumped by an 8½ by 10 in. Aldrich pump (with a Campbell & Kelly steel-lined centrifugal pump as a reserve) to the No. 1 air agitator. The flow is continuous through the first battery of six agitators—each 15 ft. in diameter by 45 ft. deep, with 55° cone, and having a capacity of 6,000 cu. ft. Agitation is effected by a central air lift. From the No. 6 agitator the pulp is elevated by an air lift to a height of 6 ft. above the tank and delivered to a system of launders, where it is further diluted with about 4 parts of partly precipitated solutions, and thence flows to four 30 by 12 ft. Dorr thickeners to be again thickened, the overflow going by gravity to the precipitation supply tanks. The thickened pulp is delivered by air lifts to the No. 7 agitator and flows continuously through the second battery of six agitators to the 28 by 20 ft. filter stock tank, equipped with a Trent agitator to keep the pulp from settling. The total period of agitation, under normal conditions, is about 48 hr.

By thus thinning the pulp with barren solution after it has passed through the No. 6 agitator and re-thickening, a large proportion of the dissolved silver and gold is displaced, which relieves the filters from this work and allows a change of solution for the final agitation.

Enough cyanide is introduced into the No. 1 agitator to bring the strength up to 6 lb. per ton of solution; about 0.18 lb. of lead acetate—a little over half the total quantity used in the mill—is also added.

About 2 per cent. better extraction is obtained with heated solutions than with cold, and, therefore, the solutions in the agitators are heated to a temperature of about 90° F. by means of steam coils, placed in the central column so that the ascending pulp may keep them free from baked mud.

Filtering and Discharging Tailings

Filtering is accomplished by a 250-leaf vacuum-filter plant, constructed on the half-gravity system. That is, filling and emptying of the stock, wash solution, and wash water are effected by gravity through 16-in.

supply pipes, and the various mediums are returned by a centrifugal pump and bucket elevator.

Effluent solutions are sent to precipitation or circulating storage by two 14 by 14 in. Gould duplex wet-vacuum pumps. The final residue is discharged by gravity through the tailings flume to the slime pond.

Precipitation

All pregnant solution to be precipitated is delivered to a 30 by 10 ft. vat, called the press solution vat, from which it is pumped by two 4-in. Krogh chain-driven centrifugal pumps through three Merrill sluice-bar type clarifying presses.

The clarified solution from these presses flows through solution meters to partial-precipitation or complete-precipitation supply tanks.

Only enough solution is completely precipitated to take care of the final washing step at the filters. The rest is precipitated to a value of about 10c. per ton and used as dilution at the upper Dorr thickeners, for table wash solution and tube-mill feed solution.

Solution to be partly precipitated is pumped by a 10 by 10 in. Aldrich chain-driven triplex pump, zinc dust being added to the pump suction, through a 6-in. pipe line to three 48-frame Merrill triangular presses in the refinery at the head of the mill.

Solution to be completely precipitated is handled in a like manner by one 6 by 7 in. Aldrich triplex pump to one 48-frame Merrill press in the refinery.

Two vats to receive solutions from the presses by gravity are situated at the head of the mill, just below the refinery.

Refining

The precipitation presses are cleaned every 15 days and the product briquetted with the following fluxes (average for 1 year): Borax 2.45 per cent., soda 6.05 per cent., and sand 6.0 per cent.

The melting is done in two double-compartment, carborundum-lined Rockwell furnaces.

During the fiscal year Mar. 1, 1914, to Feb. 28, 1915, we melted 272,-809 lb. of product, which averaged, as it came from the presses, 80.6 per cent. fine in gold and silver.

The following table is an analysis of the completely precipitated product, which has a much higher base content than the partly precipitated product:

Analysis of Completely Precipitated Product

	Per Cent.
Au and Ag.....	74.23
NaCl and KCl.....	1.00
MgO.....	Trace
CaCO ₃	3.60
FeSO ₄	0.26
Al ₂ O ₃	0.60
Pb.....	2.36
Cu.....	2.10
Cd.....	1.00
Zn.....	4.70
Se.....	1.40
SiO ₂	5.50
Al ₂ O ₃	1.40
CaO.....	0.10
Undetermined.....	1.75
100.00	

The analysis of bullion given below is from a representative sample:

Analysis of Bullion

	Per Cent.
Au and Ag.....	93.23
Pb.....	2.41
Cu.....	1.02
Zn.....	0.06
S.....	0.17
Se.....	1.80
Undetermined.....	1.31
100.00	

	Assays			Per Cent. of Extrac- tion in Each De- partment			Per Cent. of Total Extraction in Each Department		
	Au	Ag	Value	Au	Ag	Total	Au	Ag	Total
Battery heads.....	0.32	32.1	\$25.66						
Battery discharge.....	0.21	30.4	22.44						
Extraction in battery.....				34.4	5.3	12.5	34.4	5.3	12.5
Battery discharge.....	0.21	30.4	22.44						
Dorr classifier overflow.....	0.08	27.1	17.86						
Extraction in tube mills.....				62.0	10.8	20.4	40.7	10.28	17.9
Classifier overflow.....	0.08	27.1	17.86						
Table tailings.....	0.06	24.2	15.72						
Extraction by concentration.....				25.0	12.0	8.9	6.0	9.0	8.34
Table tailings.....	0.06	24.2	15.72						
Agitator heads.....	0.05	21.2	13.72						
Extraction lower thickeners.....				16.6	12.4	20.0	3.1	9.33	7.8
•Agitator heads.....	0.054	18.9	12.42						
•No. 6 agitator.....	0.02	4.85	3.31						
Extraction first six agitators.....				60.0	74.3	73.3	9.4	50.99	40.57
•No. 6 agitator.....	0.02	4.85	3.31						
No. 7 agitator.....	0.015	4.2	2.82						
Extraction upper thickeners.....				25.0	13.3	14.8	1.2	2.0	1.9
No. 7 agitator.....	0.015	4.2	2.82						
•Filter stock.....	0.014	2.8	1.96						
Extraction second six agitators.....				6.6	33.3	30.5	0.3	4.3	3.3
•Filter stock.....	0.014	2.8	1.96						
•Filter discharge.....	0.01	2.3	1.58						
Extraction in stock tank and filters.....				28.6	18.0	10.3	1.25	1.56	1.48

• Samples are averages for 30 days, taken from assay records. Silver figured at 60c. per ounce for values.

The extraction from 438,708 tons, up to Mar. 1, 1915, was 96.2 per cent. of the gold, 93.0 per cent. of the silver, and 93.9 per cent. of the money value.

The table at the foot of the preceding page shows the dissolution of both gold and silver through each step of the milling operation, figured both from assay of material as it enters the department and from original assay of ore milled.

General Remarks

The mill has concrete floors throughout. The grinding and concentrating floors drain to the lower Dorr thickeners, and the lower floors to a common sump from which solution from leaks and overflows can be pumped back into mill circuit. The initial saving effected by not laying concrete floors is false economy, I believe, as the waste bound to occur in any cyanide mill will soon exceed the cost of their installation.

The following analysis of the higher-grade ores from the Belmont mine will show the base content, which probably occurs in ores of all grades somewhat in proportion to the gold and silver content:

	Per Cent.
SiO ₂	72.00
Fe.....	2.10
Mn.....	1.60
S.....	2.60
CaO.....	3.10
Al ₂ O ₃	0.42
MgO.....	2.95
Zn.....	3.00
Ag.....	4.88
Au.....	0.06
Pb.....	1.50
Cu.....	1.09
As.....	Trace
Sb.....	0.10
Se.....	0.20
Undetermined.....	4.40
	<hr/>
	100.00

The silver occurs as silver sulphide, antimonial silver sulphide, arsenical silver sulphide, with iodine as a base, and native silver; probably somewhat in the following order:

Silver sulphide.....	argentite
Antimonial silver sulphide.....	{ stephanite
	{ pyrrargyrite
Arsenical silver sulphide.....	{ polybasite
	{ proustite
Silver iodide, chloride, bromide, and native silver.	

Tests with ore agitated 48 hr. in fresh solution, without the addition of lead, indicated a cyanide consumption of 2.8 lb., and that 1 lb. of sulphocyanide and 0.02 lb. of ferrocyanide were formed. An extraction of 92 per cent. of the values was obtained. These tests show how quickly fresh solutions become foul after contact with the ore.

In practice lead acetate to the amount of about $\frac{1}{2}$ lb. per ton is introduced to take care of the sulphur, and this tends to form insoluble lead sulphide instead of combining as a sulphocyanide. Along this line experiments have been made with litharge and lead nitrate, but the results were not equal to those obtained with lead acetate. It is figured that to some extent the zinc in the solution from precipitation combines with the sulphur, the same as the lead does. In spite of the addition of lead salts as a desulphurizer, sulphocyanide to the amount of about 2.7 lb. is carried in the mill solutions, and while this cyanide combination does not hinder the dissolution of gold and silver—further than this or any other inert salt tends to increase the viscosity of the solutions—still, it is a combination of cyanide that it is almost impossible to break up, and must therefore be considered as a consumption.

Proper alkalinity has a great deal more influence in cyanidation than is generally believed. The principal mission of lime—at least it is so considered in milling practice—is to neutralize the acidity of the ore and to do this at the first opportunity. This would naturally mean adding it to the batteries with the ore feed, but to prevent the coating of the battery screens which this method would cause, it is added as an emulsion to the battery discharge.

Early experiments on Belmont ores indicated that precipitation of the dissolved gold and silver took place during agitation with either a deficiency or an excess of alkalinity. When between 0.8 and 1.5 lb. was carried no precipitation was indicated, and as these limits are satisfactory from a settling and precipitation standpoint, they have been maintained as working limits.

About two-thirds of the total amount of cyanide used is added to the pulp flow at the No. 1 tank in the first series of agitators, and at this point the temperature of the solution is raised as much as possible, usually about 15°. The other one-third is added to the second series of agitators, but instead of "sweetening" the first tank of this series, it is added to the second tank, thereby allowing one tank in which the precipitated solution which is introduced into the second series of Dorr thickeners and between the two steps of the agitation may work, the theory being that the regenerated cyanide in this solution requires no assistance from fresh salts.

Heating the solutions to 90° temperature, in addition to increasing the ultimate extraction about 2 per cent., facilitates the coagulation of the slime, being of benefit in the settling and increasing the rate of percolation in the final filtering.

The following analysis of the circulating solution will give an idea of what salts are carried.

Total solids.....	0.7 per cent.	Per Cent.
KCN.....		0.225
KCNS.....		0.135
CaO.....		0.067
Zn.....		0.045
Cu.....		0.027
SO ₃		0.009
Cl.....		0.015
K ₄ FeCN ₆		0.010

MECHANICAL OPERATION AND COSTS

Crushing and Conveying

The crusher plant has a capacity of about 100 tons per hour and is run only one 8-hr. shift per day. It is driven by one 100-hp. motor on crushers and trommel, one 20-hp. motor on waste conveyor, one 7½-hp. motor on grizzlies and undersize belt, one 20-hp. motor on inclined belt, and one 15-hp. motor on horizontal battery-bin belt and automatic tripper, a total of 162½ hp. The actual power consumption, under full crushing load, is 80 hp.

Concaves for the No. 4 short-head crushers weigh 677 lb., cost \$88, and last two years, crushing about 75,000 tons. These concaves are run about six months, then are set out, rebabbitted, and run for six months longer, at the expiration of which time they are turned end for end and give another 12 months' like service.

Concaves for the No. 7½ K crusher last about one year and crush in the neighborhood of 150,000 tons. They weigh 1,120 lb. and cost \$202.

The No. 4 mantle weighs 536 lb., costs \$132, and has a life of two years, handling about 75,000 tons.

The No. 7½ mantle weighs 1,994 lb., costs \$352.65, and lasts 18 months, crushing about 225,000 tons.

The manganese-steel trommel screen weighs 3,776 lb., costs \$572.30, and lasts two years, handling 300,000 tons.

The consumption of wearing parts in the crusher plant is 0.0325 lb. per ton of ore, at a cost of \$0.00555.

Stamping

The 1,250-lb. stamps are made up as follows: 3¾-in. stem 550 lb., tappet 180 lb., boss head 345 lb., shoe 183 lb.

One 60-hp. motor drives 20 stamps with two 6½-in. cam shafts. The stamps drop through 6 in. 104 times per minute. The actual power consumption for each 20 stamps is 52.6 hp.

The 12 mortars are of the narrow type, with 15-in. anvil block base, and weigh 12,000 lb. each. Union Iron Works forged-steel dies 5½ in. high, weighing 159 lb., and chrome-steel shoes 9 in. high, weighing 183 lb., are used. The consumption of die iron per ton of ore stamped is 0.207 lb., and the discarded iron 0.064 lb.; the consumption of shoe iron per ton of ore stamped is 0.179 lb., and the discarded iron 0.108 lb.; making a total of 0.553 lb. per ton of ore, divided as follows: Consumed 0.386 lb., discarded 0.172 lb.

The average service for cam shafts has been 27,214 tons stamped. This life seems short and is mainly due, I think, to close feeding, and to a few shafts of very inferior quality. Still, if close feeding will increase the stamp duty by one-twentieth, the saving in power cost alone will pay for the increased breakage.

Thermit welding of cam shafts has been tried, but without great success. Two shafts were welded, at the following cost per shaft:

Labor.....	\$16.00
85 lb. thermit.....	21.60
Iron punchings.....	0.38
Wax.....	1.75
Gasoline (preheat).....	2.80
Total.....	<hr/> \$42.53

The first shaft repaired was put into service July 1, 1914, and broke, in the weld, Aug. 8, 1914. The second was put into service Aug. 8, 1914, and broke, in the weld, on Sept. 13, 1914. These two shafts were welded at the break, necessitating their being turned down to original size, which is against the advice of the thermit manufacturers. On Apr. 6, 1915, another shaft was welded from two long, discarded ends, sawed so that the weld would come in the middle box, and a boss ½ in. high and 4 in. wide left, which was taken care of in babbitting the center bearing. This shaft is still in service and may show a life really worth while. Figuring 8.88 tons per stamp day, the 30-day service from the first two shafts welded would cost about 2c. per ton of ore stamped, which, in case of breakage with no spare at hand, would be better than having 10 stamps out of commission for any length of time, and therefore thermit supplies are kept on hand for this and other emergencies.

Traveling on a track 10½ ft. above the tops of the battery posts is a 5-ton crane. Two complete cam shafts, right and left, are assembled on the cam-shaft floor, so that in case of a broken shaft, or even a broken cam which would necessitate the removal of several cams ahead in order to make repairs, the entire shaft is lifted out and a new shaft put in its place, after which the removed shaft is dressed for service and placed on the rack.

The mortar blocks contain 759.35 cu. yd. of concrete. They have a

common base, a cross-section of which is shown in Fig. 4, and are arranged in two 20-stamp blocks and two 10-stamp blocks, the 10-stamp units being on the ends.

Tube Milling

One 100-hp. motor drives two tube mills by double Morse chain drive, a 10-in. chain running from the motor to the line shaft, with a cut-off coupling to one mill and a 6-in. chain with clutch coupling to the second mill. The actual power consumption varies from 98 to 127 hp., for two mills with full load. We are well pleased with this method of transmission. The arrangement is far neater and more compact than the belt drive, and much more efficient. For milling 300,000 tons the cost per ton for chain repairs was \$0.00184.

Our first tube-mill liners were 4 by 4 by 8 in. silex blocks laid flat, in seven mills, and Campbell & Kelly (local foundry) flat white iron in the eighth mill. The average cost of lining with silex was as follows:

Silex per mill, 12,300 lb.....	\$257.68
Cement per mill, 33 sacks.....	35.80
Labor in lining.....	51.56
Labor, miscellaneous.....	50.80
Total.....	<hr/> \$395.84

This lining gave a life of eight months.

Tons ground per mill.....	15,533
Cost per ton reground.....	\$0.0583
Cost per ton milled.....	\$0.048

In 1913, Komata manganese-steel lining was installed in one mill, at a cost of \$1,784.59 (\$145.30 of which was for labor of installation). This liner ran $16\frac{1}{3}$ months and reground 27,930 tons, at a cost per ton of \$0.064. Then new ribs were installed, at a cost of \$273.04, which should allow 10 months more service, making the cost per ton reground \$0.0457.

The Belmont type of ribbed liner, made by Campbell & Kelly of hard white iron, was installed in the first mill Oct. 4, 1913, at a cost of \$710 in place. At the end of 12 months' run this mill had reground 20,484 tons, at a cost per ton of \$0.0346. At the time this paper is written, after a service of 19 months, it is figured that the entire life of the liner will be two years, making the cost per ton reground \$0.0173.

We have tried French, Newfoundland, Manhattan (local), and Danish pebbles, and have satisfied ourselves that, when power, pebble consumption, and grinding efficiency are considered, Danish pebbles are the most economical.

With Belmont ore the most efficient pebble load was found to be a

little over half full, and the highest grinding efficiency to be with 39 per cent. moisture. (Fig. 5.)

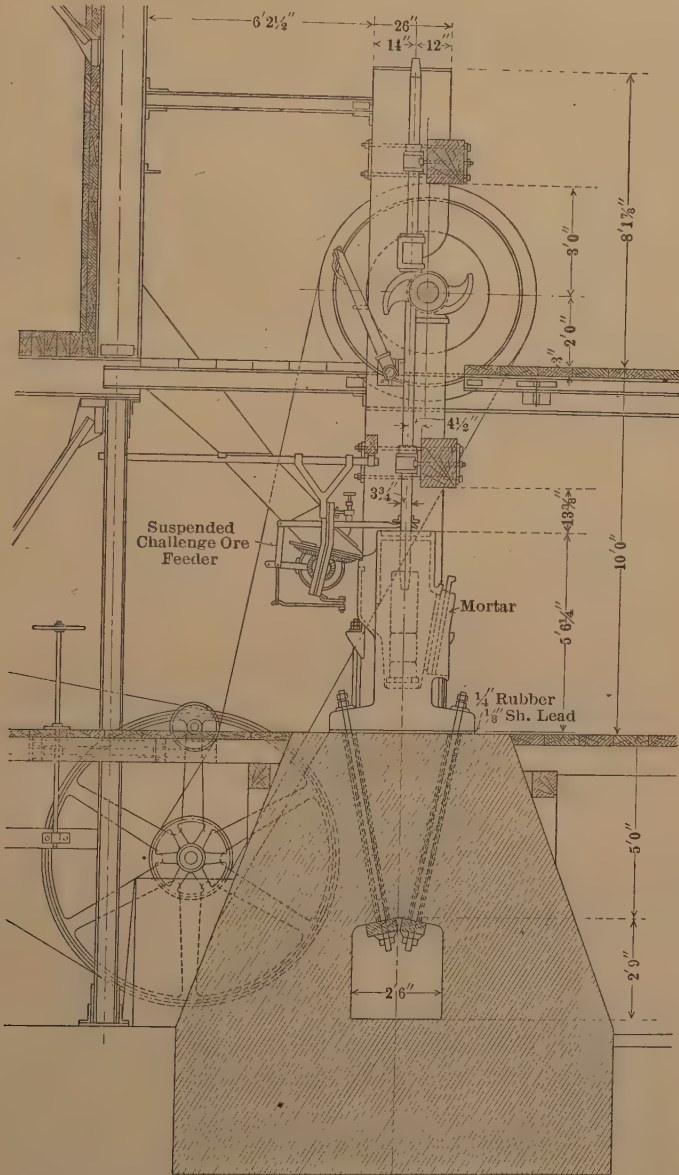


FIG. 4.—SECTION OF MORTAR BLOCK.

We are grinding at this time with seven mills about 75 per cent. through 200 mesh.

Below is given a screen analysis of the product at the different stages of grinding.

Battery Discharge—6-Mesh Tyler Ton-Cap Screens

Mesh.	Per Cent.	Cumulative Per Cent.
+ 10	7.6	7.6
+ 14	12.0	19.6
+ 20	11.1	30.7
+ 28	11.9	42.6
+ 35	8.8	51.4
+ 48	7.8	59.2
+ 65	5.2	64.4
+100	7.3	71.7
+150	6.8	78.5
+200	2.0	80.5
-200	19.5	100.0

Tube-Mill Discharge

+ 35	1.0	1.0
+ 48	3.1	4.1
+ 65	6.5	10.6
+100	20.0	30.6
+150	28.6	59.2
+200	12.7	71.9
-200	28.1	100.0

Dorr Classifier Overflow

+100	1.0	1.0
+150	12.5	13.5
+200	14.2	27.7
-200	72.3	100.0

Concentration

The 16 No. 6 Wilfley tables, running 233 strokes per minute with $\frac{5}{8}$ -in. throw, are driven by one 20-hp. motor (actual power consumption 9.38 hp.).

The tailings launder is situated in a subway under the floor, which subway also carries the wash-solution supply pipe, the water feed to the tables being through a rising pipe with gooseneck discharge into the wash box. This makes a much neater arrangement than having launders and piping above the floor.

Thickening

The eight Dorr thickeners, two of which are equipped with settling trays, are driven by one 10-hp. motor (actual power consumption 5 hp.).

The lower thickeners, which supply feed for agitation, are so arranged that by means of air lifts, which discharge a few inches above the tops of

the thickeners, the pulp is delivered to a small cone-shaped tank which acts as a suction supply for the pump delivering to the agitators. The air for these lifts is regulated by a float in the supply tank, which, in case the flow becomes too great for the pump to handle, closes the air supply until the pulp level is lowered in the tank. With this arrangement no attention whatever has had to be given to the supply of this pump during the three years' operation.

Agitation

Agitation is effected by an air lift through the central column of the agitating tanks. A 6-in. air line is carried about 7 ft. above the agitators and a $\frac{3}{4}$ -in. line dropped to each, inside the center column, discharging down through an open pipe about 18 in. above the bottom of the column. Each tank is 15 ft. diameter by 45 ft. deep and has a capacity of 6,000 cu. ft., requiring $67\frac{1}{2}$ cu. ft. of free air per minute, at 30 lb. pressure, which costs \$0.00003585 per cubic foot, or \$0.0803 per ton agitated. About

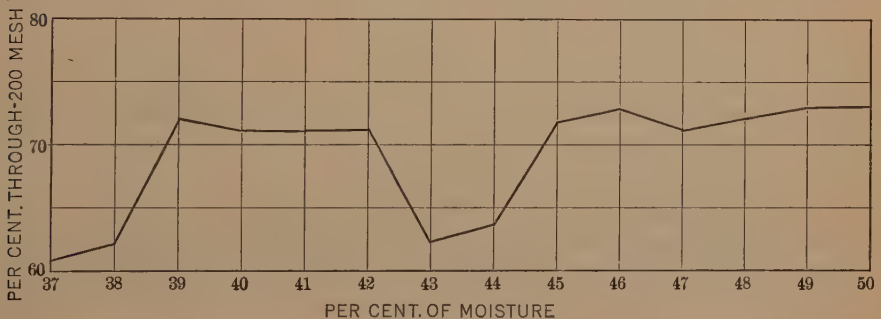


FIG. 5.

300 cu. ft. of air is used in other parts of the mill for air lifts, etc., at a cost of \$0.0485 per ton of ore treated. As previously stated, a better extraction is obtained by heating the solutions, and as the heating efficiency of steam at 5 lb. pressure is only 11 per cent. less than at 125 lb. pressure, our compressor is steam-driven and the exhaust steam used to heat solutions, and, incidentally, these warm solutions heat the mill more evenly, I think, than it could be heated by any other means. Figuring this way, the total cost of heat and air is \$0.1288, as against the cost of air alone from an electrically driven compressor (at \$0.000024 per cubic foot) of \$0.076, making the charge for heating \$0.0528, or less than half of what this charge would be with a double installation. Moreover, we have air that never dies, a very great advantage when the electric power is off, as continuous agitation is insured and settling in the agitators is avoided.

Vacuum is produced by two 14 by 14 in. Gould duplex wet-vacuum pumps, driven by one 20-hp. motor (actual power consumption 11.39

hp.). Connected with this line shaft through a friction clutch is a 15-hp. steam-driven turbine engine, which is used, in case the electric power fails, to maintain vacuum until the charge in process can be finished.

Filtering and Discharging

Filling and emptying at the filters are accomplished by gravity. Solutions and pulp are returned by steel-lined centrifugal pumps and bucket elevator, with a total motor equipment of 60 hp., divided as follows: Stock return elevator 15 hp. (actual power consumption when running 8.17 hp.), stock-return pump 15 hp. (actual power consumption when running 11.2 hp.), wash-solution return pump 15 hp. (actual power consumption when running 9.38 hp.), and wash-water return pump 15 hp. (actual power consumption when running 7.3 hp.).

The following table of costs of elevating wet pulp with both bucket elevator and centrifugal pump may be of interest:

Cost of Elevating Filter Stock Pulp 55 ft. with 20-in. Bucket Elevator .

149 ft. of 20-in. Balata belt at \$2.80.....	\$392.70
36 buckets at \$2.90	104.40
220 elevator bolts.....	20.24
7½ lb. washers.....	0.83
4½ ⁵ / ₁₆ -in. shaft.....	22.22
2½ ⁵ / ₁₆ -in. shaft.....	4.36
Babbitt.....	12.30
Labor to install new belt.....	193.41
Repair labor.....	107.12
Power at \$8.82 hp.-month, 7.3 hp.....	772.56
Chain drive to be installed.....	104.85
<hr/>	
Total.....	\$1,734.99
Wet tons elevated.....	1,045,162
Cost per ton.....	\$0.00165

Cost of Elevating Filter Stock 55 ft. with 4-in. Solid-Lined Krogh Pump

Liners.....	\$482.57
Runners.....	126.72
Housings.....	81.00
Bushings.....	36.00
Shafts.....	22.84
Chain and sprocket.....	47.65
Gland.....	1.89
Strand.....	11.25
Labor.....	53.12
Power at \$8.82 per hp.-month, 39 hp.....	408.00
<hr/>	
Total.....	\$1,270.68
Wet tons pumped.....	755,255
Cost per ton.....	\$0.00175

Precipitation

Solutions to be clarified are pumped through three Merrill sluice-bar type presses by a 4-in. Krogh chain-driven centrifugal pump, driven by a 10-hp. motor (actual power consumption 5.52 hp.), with a like installation for reserve.

These presses have a maximum capacity of 50 tons each per hour, and actually handle, on an average, 40 tons per hour. They are sluiced 12 times every 24 hr., this operation taking about 6 min. No. 12 Ames-Harris-Neville Co. duck ($9\frac{1}{4}$ to $9\frac{1}{2}$ oz.) is used, and lasts from 70 to 100 days, depending on the amount of suspended matter in solutions to be clarified. For covering one press, 63 yd. of 36-in. duck, costing 27c. per yard, is required, and the labor cost is \$7.

5 presses covered at \$24.50.....	\$122.50
150 days run at 3,000 tons per day, or 450,000 tons clarified.	
Cost per ton clarified.....	0.00027
Cost per ton of ore treated.....	0.0017

Clarified solutions to be partly precipitated, together with the required amount of zinc dust, are fed to the suction of one 10 by 10 in. chain-driven triplex pump and sent through three 48-frame Merrill presses in the refinery at the head of the mill. Solutions to be completely precipitated are handled in the same manner by one 6 by 7 in. triplex pump through one 48-frame press at the refinery.

Two circuits of precipitated solutions are maintained, for economic reasons, as 100 per cent. efficiency in precipitation, as in any other work, necessitates the expenditure of very much more energy and material than is called for by a lower percentage of efficiency, and only enough solution is completely precipitated to take care of the final wash at the filters, and the rest precipitated to from 10c. to 15c. per ton. By this method 1 oz. of bullion was precipitated (average for the past fiscal year) for 0.7808 oz. of zinc dust.

Fiscal year Mar. 1, 1914, to Feb. 28, 1915

171,894 lb. zinc used, or 2,506,214.00 Troy ounces.	
Fine bullion melted	3,209,662.36 Troy ounces.
Zinc consumption 0.7808 oz. per ounce of bullion.	

The precipitated-solution tank below the refinery is fitted with a 16-in. stand pipe, extending from the bottom of the tank to a height of 2 in. above the solution level, and from the lower end of this pipe a 4-in. line leads to the filter wash storage. Into this stand pipe the completely precipitated solution flows, and if not needed for wash at the filters overflows into the tank proper, into which the partly precipitated solution from the refinery flows. This partly precipitated solution is used for dilution

at the upper Dorr thickeners, etc., all that is not needed for this purpose overflowing from the solution tank to the battery storage tank, there joining the circulating solution from the mill, to be used in stamping. By this arrangement only one telltale is required, placed on the battery tank.

Life and Cost of Cloth Covering for Precipitation Presses

Material	Yards per Press	Cost per Yard	Life	Cost per Press	Cost per Ounce Bullion	Cost per Ton Ore Treated
No. 10, 50-in. duck ..	68	\$0.41	6 mo.	\$27.88	\$0.00006	\$0.00123
55-in. sheeting	68	\$0.19	1 mo.	\$12.92	\$0.00016	\$0.00342

Refining

The product from the precipitation presses is dumped by hand into steam-jacketed steel cars, run directly under the presses. The loaded cars are switched to one side of the refinery, where they are connected to the steam supply pipe and dried over night to about 15 per cent. moisture. After weighing, and adding the proper fluxes, the product is briquetted in a Grath Little Giant brick machine. Two special dies $3\frac{1}{2}$ in. in diameter are used, making bricks from $2\frac{1}{2}$ to 3 in. thick, depending on pressure and feed, and weighing about 2 lb. each. In practice we are able to briquet 1,000 lb. per hour.

Melting is done in two double-compartment Rockwell furnaces, lined with carborundum, and heated with crude oil and air under about 14 oz. pressure. The ordinary melt of about 150,000 oz. is gotten out in less than 36 hr. after the fires are lighted.

Lining Cost for Rockwell Furnaces. Five Compartments Lined. (Fiscal Year 1914-15)

Labor to line.....	\$80.00
2,765 lb. carborundum screened and reused (labor) ..	21.00
1,500 lb. of carborundum kaolin.....	80.00
10,735 lb. carborundum.....	554.00
4 bbl. water glass	85.00
Total.....	\$740.00
Fine ounces gold and silver melted.....	3,209,662.36
Cost per 1,000 oz.....	\$0.2305

Over the Rockwell furnaces there are steel hoods, connected with a horizontal dust chamber running the full length of the refinery, 80 ft. 3 in. At one end of the refinery, connected with this dust chamber, is a flue, which runs down 20 ft., at an angle of about 45° , to the ground, thence 49 ft. up the hill, at an angle of about 20° , to a 20-ft. vertical stack.

The percentage of gold and silver recovered from this flue, for the past fiscal year, as shown in the following table, was only 0.007 per cent. of the total bullion melted.

Values Recovered from the Refinery Dust Flue

Date	Weight, Lb.	Assay per Ton		Contents Recovered	
		Au	Ag	Oz. Au	Oz. Ag
4/10/14	50	15.2	1,528.0	0.38	38.2
4/26/14	52	12.0	1,246.0	0.312	32.4
	54	4.8	724.8	0.129	19.5
10/10/14	155	4.8	667.0	0.372	51.6
1/28/15	120	10.0	1,115.6	0.60	66.9
Total.....				1.793	208.6

Fine ounces melted, 3,209,662.36

Recovered from flue, ounces, 210.39

Recovered from flue 0.0067 per cent. of total bullion melted.

From the following table of assays of this dust, taken at different places, it will be seen that the higher values were nearest the hoods and that the dust at the base of the stack was of very low value and small in quantity, which leads us to believe that the loss is inconsiderable.

	Oz. Au	Oz. Ag
Assay from chamber directly over hoods.....	18.8	1,858.2
Assay from base of stack.....	2.6	677.4
Assay general.....	15.2	1,528.0

The costs given below are not the entire refinery costs, and are shown in order to give a better idea of the actual melting costs with Rockwell furnaces.

Labor and Material Costs of Operating Rockwell Melting Furnaces

105,676 oz. melted in 36 hr. from time fires were lighted, in two double-compartment

No. 3 Rockwell furnaces.

1,139 gal. oil at \$0.0383	\$43.62
Labor.....	40.95
Power for blower (20 hp.).....	6.00

Total..... \$90.57

Cost per ounce, \$0.0009

Labor segregated:

Head melter, 2 shifts at \$175 per mo.	\$11.70
Melters, 2½ shifts at \$4.50 per shift.....	11.25
Helpers, 4½ shifts at \$4.00 per shift.....	18.00

\$40.95

The refinery slag is ground in a 6-ft. ball mill and concentrated over a half-size Wilfley table. The concentrates, which carry most of the metallic content, are briquetted with the product for the next melt, and

Milling Costs for the Fiscal Year 1914-15, per Ton of Ore Treated

	Labor	Supplies	Power	Total
Crushing and conveying.....	\$0.036	\$0.020	\$0.010	\$0.068
Stamping.....	0.056	0.155	0.096	0.307
Classifying.....	0.014	0.002	0.006	0.022
Tube milling.....	0.032	0.102	0.230	0.364
Thickening.....	0.024	0.028	0.003	0.055
Concentrating.....	0.028	0.042	0.008	0.078
Agitating slime.....	0.050	0.764	0.007	0.821
Filtering and discharging.....	0.061	0.083	0.027	0.171
Precipitation.....	0.028	0.086	0.020	0.134
Assaying.....	0.009	0.005	0.000	0.014
Refining.....	0.017	0.024	0.002	0.043
Lighting.....	0.001	0.001	0.008	0.010
Shift bosses.....	0.036	0.000	0.000	0.036
Watchman.....	0.012	0.000	0.000	0.012
Surface and plant.....	0.015	0.006	0.000	0.021
Totals.....	\$0.419	\$1.318	\$0.419	\$2.156

Segregation of Supplies Used

	Quantity per Ton	Cost per Ton
Hydrochloric acid.....	0.17 lb.	\$0.00780
Belting.....		0.00336
Cyanide.....	3.52 lb.	0.60100
Conveyor parts.....		0.00040
Crusher parts.....		0.01260
Lead acetate.....	0.366 lb.	0.03300
Laboratory supplies.....		0.00300
Lime.....	5.47 lb.	0.04100
Fuel oil.....	2.5 gal	0.10900
Pump parts.....		0.01100
Pebbles.....	4.16 lb.	0.05700
Tube-mill lining.....		0.02300
Zinc.....	0.952 lb.	0.07200
Refinery fluxes.....		0.00800
Battery shoes.....	0.287 lb.	0.01350
Battery dies.....	0.271 lb.	0.01480
Water (other than mine).....		0.16500
Miscellaneous.....		0.14254
Total.....		\$1.31800

the tailings from concentration, averaging from \$40 to \$50 per ton in value, are shipped to the smelter. During the first few months of operation these slag tailings were fed again into the mill circuit at the batteries, but a few months of this practice resulted in such an accumulation of copper that it was discontinued in favor of the present method.

The following tabulation shows the rated horsepower, input under full load, and the average monthly consumption of the various motors throughout the mill:

Department and Duty	Rated Horse-power of Motor	Power Consumption while Running		Ave. Hours Run per Day	Monthly Average
		Kilowatts	Horse-power		
Crushing:					
Crushers and trommel.....	100	33.90	45.44	8	15.146
Waste conveyor.....	20	8.50	11.39	8	3.796
Waste-conveyor extension.....	25	14.00	18.76	8	6.253
Grizzly.....	7½	3.00	4.02	8	1.340
Inclined conveyor.....	20	8.00	10.72	8	3.240
Horizontal conveyor.....	15	6.00	8.04	8	2.680
Stamping:					
20 stamps.....	60	39.25	52.60	24	52.600
20 stamps.....	60	39.25	52.60	24	52.600
20 stamps.....	60	39.25	52.60	24	52.600
Classifying:					
8 duplex Dorr.....	20	6.50	8.71	24	8.710
Tube Milling:					
Nos. 1 and 2 tubes.....	100	87.90	117.80	24	117.800
Nos. 3 and 4 tubes.....	100	95.30	127.70	24	127.700
Nos. 5 and 6 tubes.....	100	82.20	118.00	24	118.000
Nos. 7 and 8 tubes.....	100	73.60	98.67	24	98.670
Concentrating:					
16 No. 6 Wilfleys.....	20	7.00	9.38	24	9.380
Thickening:					
8 Dorr thickeners.....	10	3.73	5.00	24	5.000
Circulating:					
10 by 12 triplex pump.....	30	15.00	20.10	24	20.100
6-in. centrifugal pump.....	30	12.50	16.70	Reserve	
Agitating:					
8 by 10 triplex pump.....	15	6.25	8.37	1	0.697
8 by 10 triplex pump.....	15	6.25	8.37	1	0.697
4-in. centrifugal pump.....	15	9.50	12.75	24	12.750
Lime mixer.....	5	1.00	1.34	24	1.340
Freight elevator.....	5	5.25	7.03	2	0.585
Filtering:					
14 by 14 vacuum pumps (2).....	20	8.50	11.39	24	11.390
6-in. centrifugal pump on Trent agitator.....	15	6.50	8.71	24	8.710
4-in. centrifugal pump, wash-water return.....	15	5.50	7.37	16	4.910

Department and Duty	Rated Horse-power of Motor	Power Consumption while Running		Avge. Hours Run Per Day	Monthly Average
		Kilowatt	Horse-power		
Filtering:					
4-in. centrifugal pump, solution wash return.....	15	6.08	8.17	14	4.760
4-in. centrifugal pump, stock return.....	15	7.5	9.38	10	3.900
Mechanical mixer.....	10	3.25	4.35	19	3.430
20-in. elevator.....	15	8.34	11.18	19	8.840
Precipitating:					
10 by 10 triplex pump.....	30	13.31	17.56	24	17.560
6 by 7 triplex pump.....	15	5.83	7.81	24	7.810
Zinc feeders.....	5	2.0	2.68	24	2.680
4-in. clarifying centrifugal pump..	10	4.12	5.58	24	5.580
4-in. clarifying centrifugal pump for reserve.....	10	4.12	5.58
Assaying:					
Crushers.....	15	5.0	6.7	4	1.116
Hot plates.....	5.0	6.7	6	1.675
Refining:					
Grinding and concentrating slag and briquetting.....	15	6.25	8.37	2	0.870
Blower.....	20	6.91	9.26	2	0.965
Machine Shop:					
General.....	20	7.5	10.72	12	5.360
Lighting.....	12.450
Total.....	813.690

Horsepower per ton milled, 1.627.

Average for year, 1.68.

Cyaniding Practice of Churchill Milling Co., Wonder, Nev.

BY E. E. CARPENTER,* E. M., WONDER, NEV.

(San Francisco Meeting, September, 1915)

BELIEVING that the results accomplished in the mill of the Churchill Milling Co., Wonder, Nev., during the fiscal year ended Sept. 30, 1914, will be of interest, I am presenting the more prominent facts and figures showing the progress made in treating this ore with especial reference to tonnage, supplies, costs, and extractions.

The mill is situated at Wonder, Nev., in the Wonder mining district, 55 miles east from Fallon, Nev., the nearest railroad point. All supplies are hauled this distance by freight teams, this method of transportation having been found more reliable for continuous service than trucks, the price paid for hauling being slightly under one cent per pound. In this mill are treated the ores of the Nevada Wonder Mining Co.

Ore is Unusually Hard

The company which tested this ore before the mill was built reported "This ore is an unusually hard, tough, oxidized quartz, carrying a small percentage of high-grade sulphides, containing good values in both gold and silver. None of the rebellious silver minerals appear to occur in appreciable quantities." . . . "In regard to the mill tonnage this ore is unusually hard and tough, and a high stamp duty cannot be expected. Using a 20-mesh screen and 1,050-lb. stamps, our best work was three tons per stamp per 24 hours."

It is very probable that the ore delivered to the testing plant was harder than the regular mill run has proved to be. However, at present the mine is developed to a depth of 1,000 ft., and still shows hard ore in places at this depth in about the same proportion as on the upper levels from which the test ore came; during this year's work, ore was treated from nearly all levels.

Together with this hard ore in the mill feed a considerable proportion appears to be broken down and tending to crumble, thus making an ideal mixture for stamp milling. A small proportion of talc is also noticeable, which sometimes increases in amount to the point where it is very troublesome in the thickeners.

* Superintendent, Churchill Milling Co.

Mill Capacity 133 Tons

The mill was designed and built as a 75-ton mill and cyanide plant on a 30° slope, the equipment being as follows, named in order of use:

- 1 10 by 16 in. Blake crusher.
- 10 1,400-lb. stamps.
- 1 6 ft. Trent Chilean mill.
- 1 Dorr duplex classifier.
- 1 5 by 22 ft. tube mill.
- 1 24 by 14 ft. Dorr thickener.
- 4 15 by 45 ft. Pachuca-type agitators.
- 1 28 by 10 ft. Dorr thickener.
- 1 28 ft. by 14 ft. stock tank with stirrer.
- 2 8 by 11 ft. 6 in. Oliver filters.
- 5 Zinc boxes.
- 2 Melting furnaces.

The necessary equipment of solution pumps, slime pumps, vacuum pump, boiler, compressor, gold tanks, battery storage tank, sump tank, etc., as shown in accompanying flow sheet.

The principal changes made since the mill was built are the addition of one 28 by 10 ft. Dorr thickener, between the agitators and stock tank, required to secure sufficient replacing of solution; one 14 by 11 ft. 6 in. Oliver filter made necessary by the greater tonnage and the tale; and two additional zinc boxes. Later a larger compressor was installed, the old one being held in reserve.

With the exception of these changes and the discontinuation of concentration, which was practiced only for a short time, the mill is being operated practically as built. The tonnage has been gradually increased until this year it has averaged 133.1 tons per day or 13.31 tons per stamp per 24 hr. The stamps dropped 92.4 per cent. of the time. On the basis of actual operating time, the rate of crushing was 14.40 tons per stamp per 24 hr. for the year. In September, the tonnage of 4,335 tons for 30 days averages 144.5 tons per day, and, based on actual operating time, 15.45 tons per stamp per 24 hr.

The annual report published in November, 1914, shows that the ore contained 10,515.895 oz. of gold of which 10,005.055 oz. were shipped as bullion; 946,050.49 oz. of silver of which 874,085.49 oz. were shipped as bullion (except for one refinery-slag shipment); showing a recovery of 95.1 per cent. of the gold contents, 92.4 per cent. of the silver contents, and 93.2 per cent. of the gross value. The above figures give the average contents of the mill heads as 0.216 oz. of gold and 19.48 oz. of silver.

The tonnage was arrived at by weighing each car, containing approximately 4 tons, as it came to the mill, and deducting the daily moisture determination, which averaged about 3 per cent. The precious-metal content of the ore, as shown above, was determined by combining the

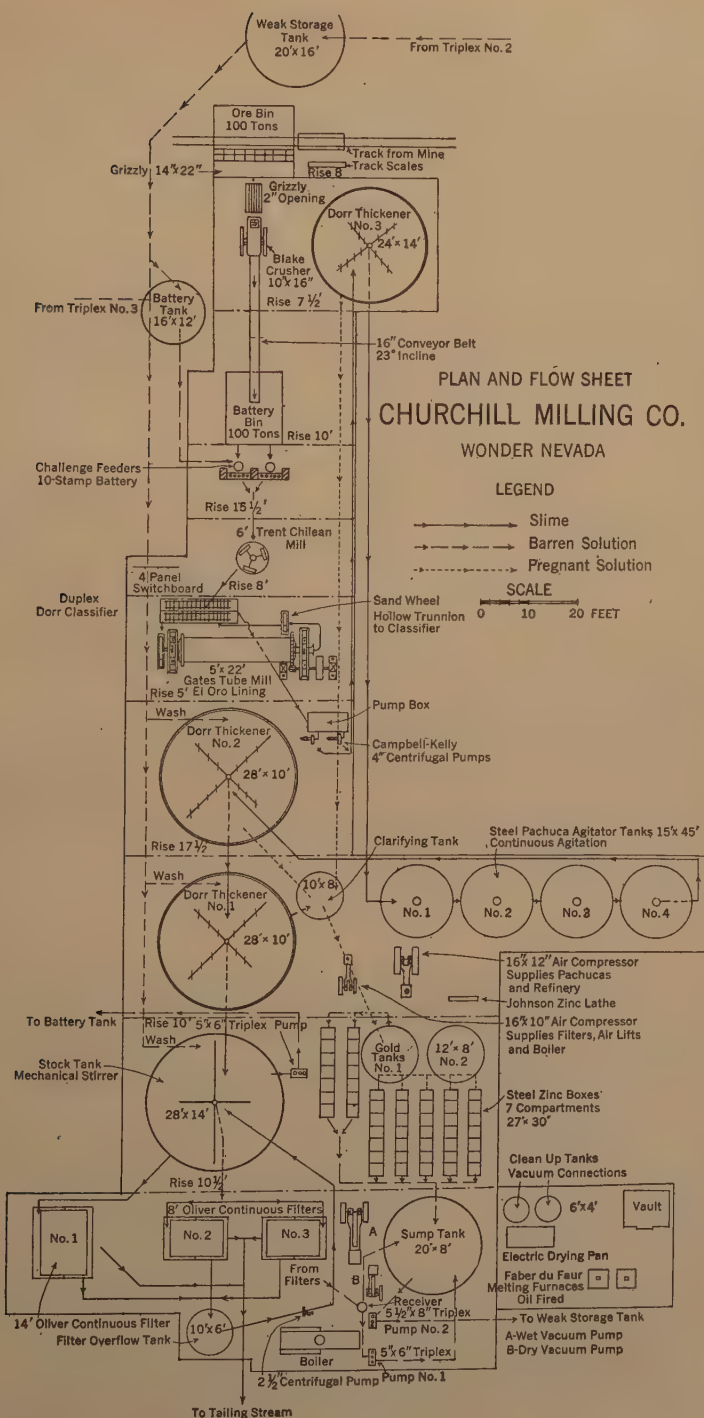


FIG. 1.—FLOW SHEET OF CYANIDE PLANT, CHURCHILL MILLING CO.

bullion shipped and the loss in tailings, the former being smelter returns and the latter the product of the daily tonnage multiplied by the average value of three separate unwashed assays of the tailings for each day, one taken from each filter. Each sample represents the cake discharged from a filter and is the result of a sample taken each half hour from each filter as the discharged cake falls into the sluicing launder; this cut is taken in a dipper which insures catching a proportionate amount of water.

TABLE I.—*Tonnage, Cost, and Recovery Data*

Months	Tons per Month	Tons per Day	Total Direct Cost	Cost to Date	Gold Recovery, Per Cent.	Silver Recovery, Per Cent.	Value Recovery, Per Cent.
Oct., 1913.....	4,160	134.2	\$2.825	\$2.825	95.3	92.8	93.6
Nov., 1913....	4,032	134.4	2.852	2.838	94.2	92.9	93.7
Dec., 1913.....	4,103	132.4	2.812	2.830	94.3	93.4	93.7
Jan., 1914.....	3,357	108.3	3.189	2.906	95.2	92.1	92.5
Feb., 1914.....	3,254	116.2	2.861	2.898	93.8	93.6	93.7
March, 1914...	4,350	140.3	2.567	2.836	94.8	93.7	93.9
Apr., 1914.....	3,730	124.3	2.604	2.804	95.3	92.2	93.1
May, 1914.....	4,217	136.0	2.598	2.777	94.3	91.1	91.2
June, 1914....	4,200	140.0	2.521	2.746	95.6	89.4	91.7
July, 1914.....	4,392	141.7	2.781	2.750	96.3	90.6	92.1
Aug., 1914.....	4,440	143.2	2.657	2.739	97.3	93.4	94.3
Sept., 1914....	4,335	144.5	2.771	2.743	97.3	92.7	94.0
Averages or Totals.....	48,570	133.1	2.743	2.743	95.1	92.4	93.2

Average contents of the ore: Gold, 0.216 oz.; silver, 19.48 oz.; valued at \$15.52 per ton.

Table I shows the tons treated each month, the tons per day, the total direct cost, the cost to date for each month, the gold, silver, and value recovery, together with the averages. The direct costs include all charges for labor both in and about the mill, superintendent's salary, mill assaying, storehouse expense, storing tailings, etc.; all charges for supplies including water, chemicals, repairs, replacements, and small items for permanent improvements, and the power cost as measured coming into the transformers where the current is reduced from 55,000 to 440 volts.

This table shows some interesting results in that for only one month of the year did the gold recovery fall below 94 per cent., for two months it reached 97.3 per cent. and this on an ore containing 88.9 oz. of silver for each ounce of gold. The gold recovery varied from 93.8 to 97.3 per cent.; the silver recovery from 89.4 to 93.7 per cent.; the value recovery was from 91.2 to 94.3 per cent.

TABLE II.—Average Costs for Year per Ton of Ore Treated

	Labor	Supplies	Power	Total
Superintendence.....	\$0.074	\$0.074
Crushing and conveying.....	0.045	\$0.024	\$0.012	0.081
Stamps.....	0.081	0.031	0.073	0.185
Chilean.....	0.026	0.053	0.063	0.142
Tube mill.....	0.029	0.119	0.107	0.255
Elevating and separating.....	0.012	0.016	0.010	0.038
Agitation ^a	0.148	0.589	0.133	0.870
Filtering.....	0.079	0.040	0.026	0.145
Precipitation.....	0.089	0.224	0.313
Refining.....	0.049	0.125	0.002	0.176
Assaying.....	0.043	0.022	0.001	0.066
Surface and plant ^b	0.051	0.014	0.014	0.079
General.....	0.005	0.005	0.010
Storehouse.....	0.046	0.002	0.048
Water.....	0.045	0.026	0.071
Steam heating ^c	0.035	0.155	0.190
Total.....	\$0.857	\$1.445	\$0.441	\$2.743

^a Chemicals charged in supplies for agitation were

Cyanide.....	\$0.375
Lime.....	0.110
Lead.....	0.089
	<hr/> \$0.574

^b Includes expense of storing tailings.

^c Cost of crude oil was 10.7c. per gallon at the mill.

Table II shows the direct costs divided into the various departments, and each department divided into cost for labor, supplies, and power; also the yearly average cost for each department, and the yearly average cost per ton for labor, supplies, and power. The cost for agitation appears high by reason of the fact that all chemical consumption (cyanide, lead acetate, and lime) is charged in the supplies under this heading. The total cost for these chemicals was \$0.574 per ton, leaving the other supplies charged to agitation amounting to \$0.015 per ton.

The wage scale, on an 8-hr. day, is as follows:

Solution men.....	\$150.00 per month
Solution helpers.....	4.00 per day
Refinery man.....	150.00 per month
Refinery helper.....	4.00 per day
Battery men.....	4.50 per day
Crusher men.....	4.00 per day
Repair men.....	5.00 per day

Having thus outlined the work, costs, and extractions, it may be interesting to analyze the various departments or divisions. A reference

to the flow sheet will show clearly the course of the ore through the mill and the flow of the solutions.

Blake Crusher for Preliminary Work

The ore is delivered to the mill in 4-ton side-dumping cars, each car being weighed before being emptied into the crusher bin. Over this bin are placed rails to prevent any boulders which cannot pass an 18-in. square opening from reaching the bin; from this bin the ore passes over a 2-in. grizzly to a 10 by 16 in. Blake crusher. The discharge from this crusher and the fines which passed through the grizzly are carried by a 16-in. belt conveyor to the battery bins. The total cost of crushing is \$0.081 per ton, of which \$0.045 is for labor; the largest item under supplies is the steel for the crusher, the hardness of a part of the ore causing excessive wear, and extreme fine crushing being necessary.

Stamps, Chilean Mill, and Tube Mill for Secondary and Final Crushing

Stamps, Chilean mill, and tube mill complete the crushing and fine-grinding equipment. The stamps, fed by suspended Challenge feeders, weigh 1,400 lb. each, dropping 96 times per minute through $6\frac{1}{2}$ in., in a narrow mortar. For a crushing capacity averaging 133 tons per day (to compensate for time lost for repairs, replacements, etc., this often reached 180 tons per 24 hr.) screens of 0:120 wire with $\frac{3}{8}$ -in. square opening in the clear were used. Occasionally for several days at a time screens with $\frac{1}{2}$ -in. square opening would be used, and less frequently finer screens, depending entirely on the character of the ore.

The shoes are 9 in. in diameter and the dies $9\frac{1}{4}$ in. in diameter; the stems are of mild steel 4 in. in diameter; the cam shaft, of hammered iron, is $6\frac{15}{16}$ in. in diameter. Very little trouble is experienced from broken stems, and the original cam shaft, installed when the mill was built, is still in commission, having been doing duty for over three years, crushing a total of approximately 130,000 tons of ore. The cam-shaft boxes are without caps and not babbitted, the one nearest the cam-shaft pulley being 18 in. long, while the others are 15 in. in length.

Chrome-steel shoes and dies were used throughout the year and the following table gives the consumption of steel per ton of ore, and the cost:

	Pounds
Shoes.....	0.232
Dies.....	0.107
Total.....	0.339
Total cost for steel per ton of ore.....	\$0.018

The ore is crushed in solution in the batteries and no attempt is made at classification between the batteries and the Chilean mill, all the battery discharge being fed direct to the Chilean, which is a Trent 6-ft. mill revolving 30 times per minute.

The screens on the Chilean vary considerably on account of the different tonnage going through the mill, but the standard used most of the time is a "ton-cap" screen No. 693. The drive on this mill is a Morse silent chain drive 5 ft. between centers, which is giving excellent service. Chrome steel is also used in this mill and the average consumption for the year was 0.50 lb. per ton of ore crushed, costing \$0.034 per ton.

Following the Chilean mill and in closed circuit with the tube mill is a Dorr classifier, and the remarkable work it is doing deserves particular mention. Frequently runs of 180 tons per day have been made, all of which passed through this classifier, since it is the only one in the mill, and to this must be added at least 85 tons coming from the tube-mill discharge, making a total of 265 tons per day, and even when overloaded to this extent the only attention it ever receives is a little oiling.

The tube mill is a 5 by 22 ft. Allis-Chalmers, belt driven and making 28 r.p.m. It has an El Oro lining, and the present liner, which has been in service for one year, still has three months' wear; many of the plates are worn practically smooth except for ridges which were formerly ribs. The cost per ton for lining on the total ore milled was \$0.022 counting on a 12 months' life, but this cost will probably be reduced to \$0.016. The height of the pebble load was kept at the center line both before and after the pebbles dropped out of the lining plates, the lining becoming in effect a smooth lining. No appreciable difference could be detected in the work done nor in the pebble consumption. The consumption of tube-mill pebbles for the year was 3.39 lb., costing \$0.080, per ton of ore ground.

The following screen analyses are the averages of a large number of tests made during the year.

Battery Feed—Everything to Stamps

Mesh	Per Cent.
+ $\frac{3}{8}$ in. sq.	37.6
+ 20	38.6
+ 40	6.6
+ 60	4.4
+100	3.1
+150	2.1
+200	1.2
-200	6.4

Battery Discharge and Chilean Feed

Mesh	Per Cent.
+ 20	46.9
+ 40	14.5
+ 60	8.7
+100	7.5
+150	5.0
+200	1.5
-200	15.9

<i>Chilean Discharge</i>		<i>Tube-Mill Feed from Dorr Classifier</i>	
Mesh	Per Cent.	Mesh	Per Cent.
+ 20	9.7	+100	83.6
+ 40	16.2	+150	7.0
+ 60	14.7	+200	3.0
+100	12.8	-200	6.4
+150	9.4		
+200	6.0		
-200	31.2		

<i>Tube-Mill Discharge</i>		<i>Filter Discharge</i>	
Mesh	Per Cent.	Mesh	Per Cent.
+100	8.7	+100	4.9
+150	10.5	+150	8.7
+200	11.4	+200	10.3
-200	69.4	-200	76.1

The following is a cost analysis for the combined stamping and gerinding, including Dorr classifier, taken from Table II.

	Labor	Supplies	Power	Total
Stamps.....	\$0.081	\$0.031	\$0.073	\$0.185
Chilean.....	0.026	0.053	0.063	0.142
Tube mill.....	0.029	0.119	0.107	0.255
	<hr/>	<hr/>	<hr/>	<hr/>
	\$0.136	\$0.203	\$0.243	\$0.582

Agitation and Washing in Pachuca Tanks and Dorr Thickeners

The classifier overflow is pumped to Dorr thickener No. 3, the average specific gravity being 1.110 (approximately 5 tons solution to 1 ton ore). The overflow from this thickener flows to the gold tanks for precipitation and the thickened slime passes to the first of four Pachuca agitators for continuous agitation, each agitator being 15 by 45 ft. The average specific gravity of the slime in the agitators was 1.250 (2 tons of solution to 1 ton of ore).

I wish to call particular attention to the work of this Dorr thickener No. 3. The tank is 24 ft. in diameter and 14 ft. deep, having a surface of 452 sq. ft.; therefore, an average of 133 tons per day allows only 3.4 sq. ft. of surface per ton of ore thickened, and when the tonnage reaches 180 tons per day this is reduced to 2.44 sq. ft. per ton of ore, but undoubtedly the specific gravity of the slime delivered to the agitators on these days was below the average.

Before proceeding to the detailed figures on the agitators it will be interesting to examine into the extraction made to this point. The ore first comes in contact with solution in the batteries and in its passage through the batteries, Chilean mill, classifier, tube mill (a portion of the ore only) and the Dorr thickener the precious-metal content has been reduced from the average of 0.216 oz. gold and 19.48 oz. silver to 0.051 oz. gold and 5.42 oz. silver, showing an extraction of 76.4 per

cent. of the gold, 72.2 per cent. of the silver. I do not know of any other silver ore so amenable to cyanide that in its passage through the grinding machines and one thickener on its way to the agitators an equal percentage can be taken into solution, and I consider this an interesting and unusual silver ore for this reason. It was the intention of the designers that the overflow from this thickener No. 3 should flow to the battery storage tank for battery feed solution, little thinking that it would be the richest solution in the mill, with the single exception of the solutions in the agitators, and that it would have to be precipitated before being used again.

Unless especially mentioned to the contrary all assays quoted in this article are averages of samples taken every day in the year, except for the few occasions when the mill may have been down for 24 hr. continuously; most of the samples were taken by dips or cuts each hour:

Part of the lime supply is added to the ore as it passes from the ore bin to the Challenge feeders, the rest in agitator No. 1, and the same is true of the lead acetate; the cyanide (129 per cent. sodium cyanide) is all added to agitator No. 1 after being dissolved in a small box.

Table III shows the average assay value of the ore as it enters agitator No. 1, its value as it leaves each agitator and as it reaches the filter box, for each month of the year, the average for the year; the percentage extraction and the number of ounces extracted in each step. It will be noted that the extraction in agitator No. 4 was only 0.002 oz. gold and 0.24 oz. silver, and I am satisfied that agitation in this tank could have been discontinued and the same results obtained in Dorr thickeners No. 2 and No. 1 (see flow sheet) before the slime reached the stock tank.

The difference in value of the slime in agitator No. 4 and the filter heads (both washed samples) is 0.003 oz. gold and 0.09 oz. silver, which shows the extraction that took place in the two Dorr thickeners No. 2 and No. 1 and the stock tank in which the slime was washed after agitation. The value of the solution in agitator No. 4 varied from \$5 to \$7 per ton, depending on the value of the ore, and was reduced approximately to \$1 in the stock tank by the washings applied. These two thickeners being each 28 ft. in diameter have 4.63 sq. ft. per ton of slime settled.

The usual titration in agitator No. 1 was about 4.5 KCN and 1.6 CaO. During the cold months sufficient steam was used in agitators No. 1 and No. 2 so that at some time during the 24 hr. their temperature was between 85° and 90° F., the steam being used in pipes in the uplift column.

The specific gravity of the slime in the agitators averaged 1.250 (2 tons solution to 1 ton of ore), consequently, averaging 26 lb. of dry ore per cubic foot. The available area for agitation, after deducting for the cone and to a depth of 2 ft. from the top, is 24,440 cu. ft. in four

TABLE III.—*Extraction Table—One Year's Assays*

Ore Heads 0.217 oz. gold, 19.48 oz. silver, value \$15.52

Month	Feed to Agitator No. 1		Discharge from Agitator No. 1		Discharge from Agitator No. 2		Discharge from Agitator No. 3		Discharge from Agitator No. 4		Filter Heads	
	Oz. Au ¹	Oz. Ag	Oz. Au	Oz. Ag	Oz. Au	Oz. Ag	Oz. Au	Oz. Ag	Oz. Au	Oz. Ag	Oz. Au	Oz. Ag
Oct.....	0.062	5.97	0.037	3.69	0.025	2.12	0.016	1.40	0.013	1.17	0.010	1.10
Nov.....	0.056	4.51	0.032	2.75	0.021	1.56	0.016	1.19	0.012	0.95	0.011	0.98
Dec.....	0.060	4.23	0.034	2.71	0.023	1.68	0.017	1.27	0.014	1.02	0.011	0.92
Jan.....	0.066	4.32	0.038	2.42	0.026	1.81	0.020	1.43	0.018	1.27	0.013	1.03
Feb.....	0.058	4.19	0.035	2.79	0.019	1.54	0.015	1.31	0.015	1.08	0.011	0.96
Mar.....	0.055	5.10	0.037	3.21	0.026	2.25	0.020	1.72	0.015	1.37	0.011	1.22
Apr.....	0.070	6.11	0.038	3.50	0.026	2.53	0.019	1.96	0.015	1.63	0.010	1.47
May.....	0.047	6.00	0.026	3.20	0.028	3.48	0.018	2.44	0.013	1.89	0.010	1.88
June.....	0.036	5.71	0.023	3.42	0.015	2.63	0.011	2.05	0.008	1.80	0.007	1.80
July.....	0.030	6.75	0.022	3.64	0.013	2.54	0.008	1.97	0.006	1.76	0.006	1.81
Aug.....	0.035	5.91	0.020	2.70	0.010	1.86	0.006	1.41	0.005	1.30	0.005	1.30
Sept.....	0.033	6.29	0.022	3.36	0.012	2.32	0.007	1.83	0.006	1.75	0.005	1.50
Average.....	0.051	5.42	0.030	3.11	0.020	2.20	0.014	1.66	0.012	1.42	0.009	1.33
Extraction, per cent.....												
Extraction in tanks, per cent.....			86.200	83.70	90.800	88.70	93.600	91.50	94.500	92.70	95.900	93.20
Dissolved in tanks, oz.....			9.700	11.50	4.600	5.00	2.800	2.80	0.900	1.20	1.400	0.50
Dissolved in tanks, oz.....			0.021	2.31	0.010	0.91	0.006	0.54	0.002	0.24	0.003	0.09

Dissolved before ore reaches agitator No. 1, ounces, Au 0.166, Ag 14.06. Extraction before ore reaches agitator No. 1, per cent., Au, 75.7, Ag, 72.2.

tanks, giving a total dry weight of ore of 317.2 tons. This gives each tank a capacity of nearly 80 tons and a total period of agitation of 57 hr.

Oliver Filters Used

The filtering installation consists of two 8-ft. and one 14-ft. Oliver filters, all 11 ft. 6 in. diameter. For considerable periods the two 8-ft. filters would handle the slime, but whenever a rush of talc occurred it was found necessary to use the third. Under the former conditions the filters handled 0.23 ton, or 460 lb., of ore per square foot of canvas per day with an 18-in. vacuum. This capacity could be greatly increased by thickening the slime and raising the vacuum. A dry vacuum is used and the solution removed from the receiver between the filters and the dry-vacuum pump by a small triplex pump.

Table II gives the cost of filtering as follows: Labor \$0.079, supplies \$0.040, power \$0.026, total \$0.145. At first glance the cost for labor and supplies seems high for this type of filter, but since they require equally as much attention as the thickeners, zinc boxes, stock tank or boiler, it is only fair that a small proportion of the solution man's time and the helpers' time be charged against them. Also in the way of supplies, only about 20 per cent. of the charge was for supplies for the wheel itself, the rest being for vacuum pumps, triplex pump, slime pump for pumping overflow from filter tanks back to stock tank, etc.

A saving of 0.23 lb. of cyanide was made during this year over the preceding year with practically the same grade of ore, and I believe the greater part, if not all of it, was saved on the filters, due to changing the method of washing the cake, during the last eight months of the year. This change made it possible to wash both ascending and descending sides of the cake with water.

Formerly the method was to wash the ascending side of the cake with barren solution, and the descending side with water when the amount of solution in the mill would permit, or with barren solution when crowded, it being found that a water wash applied at all times would build up too much solution on account of the flow of water into the filter box between the wires during the time the discharged section was moving past the scraper.

Barren solution washes the precious metals out of the cake, but in a silver mill, where all solutions are necessarily high in cyanide, the cyanide loss is considerable. To prevent this loss of cyanide the usual rubber apron was placed against the cake on the ascending side and a water wash applied, the surface water thus caught being diverted to the slime-discharge launder. On the descending side an iron apron, 18 in. wide, extending the full length of the filter, was placed so that the top edge, resting on supports at the ends of the filter, scraped off just the top of

the cake; the lower edge, extending over the edge of the filter box, diverted the part of the cake scraped off (possibly $\frac{1}{16}$ in.) together with the wash to the discharge launder, but did not interfere with the regular blow-off. All that was necessary to accomplish this was to place this apron so that the top edge was just above the blow-off section and at such an angle that the slime and the water wash would flow over it and drop into the discharge launder.

This method permits the use of enough water to keep a constant sheet of water flowing over both the ascending and descending sides of the cake and does not permit any of it to enter the mill solution. It insures a more efficient wash, since it is impossible always to have barren solution for washes free of precious metals. As stated above, the saving of cyanide was 0.23 lb. for the year, and had this method been in use all of the year instead of eight months, it is reasonable to suppose the saving would have been one-half greater.

On account of adding barren solution to the stock tank and decanting same, it was necessary to thicken the slime in the filter boxes. To accomplish this, a small tank was placed below the filters, 2-in. pipes being connected in the filter box at the level at which it was desired to keep the slime and near the center of the drum. This drew off the thin slime and solution on the surface, delivering it to the small tank, from which it was pumped back to the stock tank or to the thickener just above the stock tank. This served the double purpose of keeping thick slime in the filter boxes and preventing an overflow, doing away with the float regulator on the valve controlling the supply.

Before the discharged cake and water have flowed 100 ft. down a steep hill just below the mill the cake and water have become thoroughly mixed, and at about this point a sample is taken each shift for specific gravity, titration, and assay of solution. Thus, a sample for determination of the specific gravity of the discharge stream is taken three times a day. The average of all these for seven months shows a ratio of 1.38 tons of water for each ton of ore. These samples are then allowed to settle and an equal amount of the clear water or solution from each is placed in a sample for titration and assay. The average of all titrations showed 0.62 lb. of cyanide and at the above ratio this gives a loss of 0.85 lb. of cyanide per ton of ore treated. The assays of this solution and the specific gravity are then used to compute the loss in soluble gold and silver, which for seven months was \$0.18 per ton of ore. This regular sampling was in effect during only the last seven months of the year.

Precipitation by Zinc Shavings

Zinc shavings are used for precipitation. Seven 7-compartment boxes are used, each compartment having a V-shaped bottom beneath the screen and emptying into launders which carry the precipitate to

canvas-bottomed tanks connected with vacuum for drying. No zinc is used in the first compartment and immediately after a cleanup the boxes contain approximately 444 cu. ft. of zinc shavings. Since 331,750 tons of solution were precipitated during the year, the daily tonnage amounts to 909 tons or 2.05 tons of solution per cubic foot of shavings per 24 hr.

The actual cubical contents of zinc shavings varies from 444 cu. ft. immediately after a cleanup (a cleanup is made every 10 days) to probably two-thirds of that figure just before the next cleanup. This, together with lost time, would considerably increase the tons per cubic foot of shavings, so that if an accurate estimate could be formed it would be found nearer 2.5 tons than the figure given above.

Dividing the total production by the tons precipitated shows a value of \$2.12 precipitated per ton of solution entering the zinc boxes. The zinc-box tails averaged \$0.107 per ton, giving the zinc-box heads a value of \$2.227 per ton and showing an effective precipitation of 95.2 per cent. It is natural to presume that these values varied greatly from day to day and even from hour to hour. Costs for supplies and labor are shown in Table II.

Sheet zinc is purchased in carload lots and cut in the mill with a Johnston zinc lathe. One operator will cut 700 lb. per day. The charge for labor, power, and incidentals, totals \$0.007 per pound for cutting zinc. The shavings are 0.004 in. thick; 9-gauge 36 by 108 in. sheets are used.

Precipitate Refined without Preliminary Acid Treatment

During the year the dry weight of the precipitate taken from the zinc boxes was 98,610 lb., which produced 62,608 lb. of bullion, gross; the bullion averaged 968 fine in gold and silver. Thus the precipitate contained 63.5 per cent. of bullion gross and 61.5 per cent. of gold and silver.

No acid treatment was required, except on one lot of zinc shorts accumulated during the first six months of the year. When the precipitate is taken from the vacuum tanks it is placed in an electric drying pan for drying to approximately 20 per cent. moisture. The flux finally decided upon as the most satisfactory was 7 per cent. each of borax glass, soda, and sand. Melting is done in an oil furnace burning crude oil and using Dixon No. 10 retorts.

General Résumé

The additions necessary to increase the tonnage, enumerated earlier in this article, total less than \$10,000; undoubtedly next year's operations will show a further increase in tonnage and I feel safe in predicting a treatment of 16 tons per stamp per day, counting actual operating time.

Treating this tonnage makes fine preliminary crushing necessary, as will be seen by a glance at the screen test of battery feed. This, however, adds only a small additional cost to crushing, as the total crushing cost was only \$0.074 per ton of ore for labor, supplies, and power. It is doubtful if the extra cost due to crushing to this fineness increased the cost \$0.02 per ton over what it would have cost to crush to the usual size for battery feed. This is a good illustration of the advantage of fine crushing preceding stamps.

The chemical consumption was as follows:

	Per Ton of Ore	Ounces Bullion per Pound
Sodium cyanide.....	1.72	10.56
Zinc.....	1.63	10.19
Lead.....	0.94	19.45

I find myself at a loss to explain satisfactorily the forced use of lead acetate in seemingly excessive quantities during the year's work. The total consumption for the year was 0.94 lb. per ton, but the first six months averaged only 0.74 lb., while the last six months averaged 1.14 lb. At about the time this change in consumption took place assays of the slime discharge and the slime in the agitator indicated that the extraction was falling off. Without waiting to run experimental tests, the usual steps were taken to correct conditions, the temperature of the solutions was raised, alkalinity was raised and later lowered, more cyanide was added, and then, because the lead acetate consumption had been higher than usual, the weight added was cut in two, when suddenly the assays indicated a still further drop in extraction and it was apparent the trouble had been found. Double the amount of lead acetate formerly used was now added and extra lead was added to all tanks following agitator No. 1. The effect was apparent the next day in reduced slime assays all over the mill.

Since that time numerous efforts were made to reduce the lead consumption, but it was found impossible without an excessive increase in the value of the tailings amounting to several times the cost of the lead. In connection with this use of lead acetate and in order to present all available facts I will state that no noticeable difference was apparent in the cyanide and alkaline strengths, nor in the temperatures of the solutions at this time nor later, nor was the value or character of the ore changed. It is probable one of the two following reasons was to blame for the change:

1. About this time a more completely closed circuit at the filters prevented loss of solution, due to using all water wash instead of a solution wash followed by a water wash. This failure to renew solutions may have been the cause of a higher lead consumption.

2. Sheet zinc was purchased and cut in the mill, at this time, replacing the former practice of purchasing zinc shavings, which may have had some effect upon the solutions.

The perplexing question of how extraction should be figured seems to be a live issue among the fraternity at this time, and without attempting to settle it one way or the other the effect of different methods as applied to the work in this mill can be readily shown.

As explained before, the extraction is estimated on smelter returns and tailings assays. Smelter returns were \$14.46 per ton and tailings \$1.06, giving the ore a value of \$15.52 per ton and a value extraction of 93.2 per cent. This tailings sample, assaying \$1.06, contained a certain amount of solution, since the wash on the filter was not 100 per cent. efficient, and the sample was not washed after being taken. However, a regular daily sample was cut from the stream of slime flowing to the filter each hour of the 24. This sample was washed and assayed, averaging \$0.96. Adding to this the solution loss of \$0.18, shown previously, gives a total value for tailings of \$1.14; this added to the \$14.46 per ton smelter returns gives an ore value of \$15.60, indicating a value extraction of 92.7 per cent., a difference of 0.6 per cent. from the reported extraction. This extraction would certainly be unfair to the mill, however, since it has not been determined accurately what extraction takes place as the slime cake is being washed with water, dropped from the filter and discharged. The slime is thus in contact a short time with what is in reality a weak fresh solution. Most men experienced in this work will agree that some extraction takes place.

On the other hand, suppose extraction were figured on smelter returns and the product showing the lowest assay value in the mill, namely, the filter heads; this method would give \$14.46 smelter returns plus \$0.96 or \$15.42 for a new ore value, and the value extraction on these figures would be 93.8 per cent. This is manifestly an unfair extraction figure, because the treatment was not yet completed when the filter-heads sample was taken.

It is the general practice to compute extractions by using a tailings sample secured somewhere about the filter after the cake has been washed, either before or after it has been dropped. In the work at this mill it is taken after the cake has been dropped. Therefore, the slime contains certain solution values forced back into the cake by the compressed air used for discharging. I maintain that this is a fairer sample than one taken before the cake is dropped from this type of filter, or any filter discharged by allowing water or air to enter the vacuum chamber. It must be recognized that there are several viewpoints from which to approach the question of calculating extraction, and that one should be chosen which seems to be the fairest to all concerned.

Zinc-Dust Precipitation Tests

BY NATHANIEL HERZ,* E. M., LEAD, S. D.

(San Francisco Meeting, September, 1915)

THE use of zinc dust for precipitating the precious metals from cyanide solutions is well established now in many places, and has many advantages over the shavings method of precipitation. Although much work has been done on methods of testing zinc dust, as yet no absolutely positive test has been devised to determine the value of the material. Certain facts are known, but often it happens that results on a working scale are not as expected from the laboratory tests. The purpose of this paper is to record some observations made while testing a large variety of zinc-dust samples in the laboratory.

W. J. Sharwood¹ has described standard analytical methods, which have been followed, with the changes noted. The regular determinations include zinc oxide, by solution in ammonia and ammonium chloride, lead, and precipitating efficiency. In some cases cadmium was determined by a method similar to that described by J. E. Clennell.² I will discuss the precipitating-efficiency test in detail. The procedure is as follows:

"A solution of potassium silver cyanide is prepared by dissolving 10 grams of silver cyanide (AgCN) and 5 grams of '99%' potassium cyanide in a little water and diluting up to 1000 cc. It is then adjusted by addition of a little more KCN or AgCN until the solution indicates from 0.12% to 0.15% free KCN by titration with standard silver nitrate. The titration is best made by using a 10-cc. or 20-cc. sample, adding 1 cc. of 2% potassium iodide, and a slight excess of ammonia, as the end point is then sharper. Or 15 grams of pure crystallized $\text{KAg}(\text{CN})_2$ may be dissolved in a liter of water and 1.5 grams KCN added.

"Weigh out 0.5 gram zinc dust into a 300-cc. beaker. Add a few cubic centimeters of water and stir until zinc is well mixed, then pour in 250 cc. of the prepared solution, stirring vigorously. See that all lumps are broken up, and continue stirring for fully five minutes. Stir occasionally (at least every 10 minutes) until the end of two hours from the addition of the solution. Then filter upon an 11-cm. filter, wash precipitate thoroughly, sprinkle with test lead, wrap it carefully in the paper, place in a scorifier with about 20 grams test lead, burn paper cautiously in muffle, scorify five minutes, cupel at low temperature and weigh silver. Milligrams silver obtained from 0.5 gram zinc $\times 0.0606$ = percentage precipitating efficiency."³

* Homestake Mining Co.

¹ *Journal of the Chemical, Metallurgical and Mining Society of South Africa*, vol. xii, No. 8, p. 332 (Feb., 1912); reprinted in *Engineering and Mining Journal*, vol. xciii, No. 19, p. 943 (May 11, 1912).

² *Engineering and Mining Journal*, vol. xciv, No. 16, p. 793 (Apr. 19, 1913).

³ W. J. Sharwood, *loc. cit.*

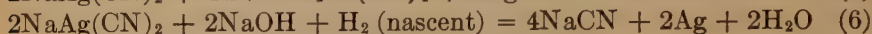
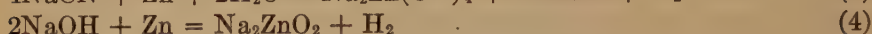
Recently I substituted the equivalent of sodium cyanide for the potassium cyanide, with no variation in the results.

A test was made with a gold solution that was equivalent to the standard silver solution. The efficiency shown was one-third to one-half that with the silver solution, in different trials. In one set of experiments a mixture of these gold and silver solutions in equal parts was used. The beads obtained were parted, and were found to contain about four atoms of silver to one of gold. The calculated efficiency was intermediate between those obtained with gold alone and silver alone, being somewhat higher than the average of the two values. This agrees with the fact that in practice gold is more difficult to precipitate than silver.

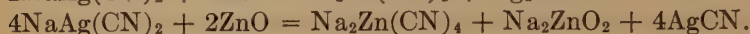
The following observations were made in the course of my experiments: Slight variations in the free cyanide present in the precipitating solution affect the results very little. However, if the free cyanide is reduced materially, to about 0.01 per cent., the results are very different. Tests agitated for short periods indicate that the rate of precipitation is much slower. In some cases exposure of the zinc to moist air for several hours so as to increase the oxide present seemed to increase the efficiency, although the amount of metal present in 0.5 g. was less. Also, in some cases, the addition of a little freshly ignited zinc oxide to the solution at the start of the precipitation increased the amount of silver precipitated. This seems to be due to the action of the oxide on carbonates in the solution. In this solution of low cyanide strength there is not enough free cyanide to dissolve all the zinc oxide, and the excess of zinc oxide would react with the alkali carbonates introduced from the cyanide used, forming zinc carbonate and caustic alkali. It is well known that the presence of caustic alkali in a solution increases the rapidity and completeness of precipitation.

As the solutions in practice are ordinarily free from carbonates, containing their alkali as lime, the standard method of preparing the silver solution was altered as follows: The solution is made up to double strength in silver. After adjusting the cyanide strength to about 0.2 per cent. NaCN, a slight excess of lime water is added to convert carbonates into hydroxides, and the solution filtered. It is then made up to the final volume, and tested for protective alkalinity. If this is not equivalent to about 0.01 to 0.015 per cent. NaOH, the solution is adjusted by the addition of a little acid. I have never found the alkalinity lower than desired, due to the alkali present in the cyanide as impurity. Although this adds to the work of preparing the solution, it is worth the trouble, as the results are more uniform, and come nearer to actual practice. Slight variations in the solution, such as the addition of neutral salts and changes in temperature, have much less effect than with the old standard solution.

The following reactions may occur:



The following two do not occur:



For all practical purposes, reaction (5) may be taken as the important one, whether it takes place by itself, or with (3), (4), and (6) as intermediate stages. Reactions (1) and (2) consume cyanide and alkali, interfering with (3) and (4), and thus indirectly interfering with (6). The free cyanide and alkali in the standard solution are sufficient to take care of more oxide than is usually found in zinc dust.

As precipitation is always better in the presence of reasonable quantities of cyanide and alkali, and is electrolytic in its nature, the addition of neutral salts as noted above would affect the solutions of lower cyanide and alkali concentrations much more than the standard solution. It is also well known that chemically pure zinc is very much less active than zinc containing a reasonable amount of metallic impurities. I will touch briefly on the common impurities.

Impurities in Zinc Dust

Zinc Oxide.—Zinc oxide is always present, varying from under 5 per cent. to over 20 per cent. Unless zinc dust containing over 15 per cent. of the oxide is unusually efficient for other reasons, it should be condemned, as the oxide only dilutes it and consumes cyanide. The oxide can rarely be seen in any quantity, even under the microscope, but the indications are that it occurs as a film on the surface of the particles, or in minute pits on the surface. It is present, sometimes in considerable quantities, in the small amount of coarse material, as well as in the finest part of the dust, so it cannot occur as separate grains disseminated through the mass of the zinc.

In regard to the oxidation of zinc dust during handling, in dry air it is practically negligible. In some tests, exposing a layer about $\frac{1}{2}$ in. deep to dry air, with occasional stirring, for several weeks, the oxide was increased only a very few per cent. On the other hand, in moisture-saturated air the oxidation may be very rapid, amounting to as much as 2 or 3 per cent. per day, but the rate falls off as the oxidation goes on, presumably due to the protection of the particles by a film of oxide. Exposure of a fairly large surface, however, for a reasonable time, does

not reduce the efficiency of the zinc as much as is commonly supposed, as only the surface layer would be oxidized, and to only a small extent. In a few cases exposure for a few hours seems to be of advantage, as will be explained later.

Metallic Impurities.—The only metallic impurities of any importance are lead and cadmium. In the samples I have examined, the lead varies from almost none to about 5 per cent., and cadmium from none to about 4 per cent. Both are probably present as metal alloyed with the zinc, as they are less easily oxidized than zinc. I know of no simple way to prove this point, but it seems self-evident in the case of lead. In the case of cadmium, a rapid extraction with the ammonia-ammonium chloride solution used for dissolving zinc oxide dissolves no cadmium, but metallic zinc would precipitate cadmium from this solution, if any were dissolved.

The nature of alloys of cadmium and lead with zinc has not been studied by modern metallographic methods, but the following statements can be made: It is known that lead is slightly soluble in zinc, and on cooling, about 1.3 per cent. of lead is left in the zinc, as a solid solution, any more lead segregating as a solution of zinc in lead. Therefore, if cadmium is absent, the metallic portion of the zinc may contain 1.3 per cent. of lead in solution, possibly in molecular form, while any excess must be in larger particles. This can be shown by preparing an alloy, and cooling it slowly, cutting out all parts showing segregations of lead, and analyzing the remainder. Cadmium is soluble in melted zinc in all proportions, and insoluble in solid zinc; but, separating only at the instant of solidification, does not segregate in large masses. Lead is soluble in melted cadmium in all proportions, and partly soluble in solid cadmium, the excess of lead not segregating as in zinc. The probability is that cadmium would increase the solubility of lead in solid as well as melted zinc, and would certainly diminish the tendency toward segregation. I prepared an alloy containing about 2 per cent. cadmium and 2.2 per cent. lead, and let it cool slowly in a muffle. It seemed perfectly homogeneous, and if there was any undissolved lead, it was so finely disseminated as to be invisible on etching the specimen.

Lead is known to increase the activity of zinc in cyanide solutions, as it has a much lower electromotive force, and remains undissolved. It is not known whether the dissolved portion (in the zinc) is more efficient than segregations, or *vice versa*. Cadmium also has a lower potential than zinc, and would probably also help, but would not be as advantageous, as the difference in potential is less. From several analyses of precipitates, the indications are that the cadmium remains in the precipitate, which has been proved to be the case with lead. It is to be hoped that the metallography of this system will be worked out soon so as to answer the question of the distribution of the impurities. The following

analyses show the variation in some samples of zinc dust from different sources:

Lead	Cadmium	Zinc Oxide	Total	Per Cent. of Metallic Portion	
				Lead	Cadmium
0.15	0.00	8.30	8.45	0.16	0.00
1.18	3.10	9.60	13.90	1.31	3.43
1.75	1.70	8.45	11.90	1.91	1.86
1.80	1.20	17.85	20.85	2.19	1.46
1.82	1.85	10.15	13.80	2.02	2.06
2.04	2.15	11.65	15.85	2.31	2.46
2.06	3.90	10.50	16.45	2.30	4.36
2.38	1.85	11.85	16.10	2.70	2.10
2.60	1.25	6.30	10.15	2.78	1.34
3.05	1.90	16.15	21.10	3.64	2.27
3.65	1.15	10.60	15.40	4.08	1.29
3.73	2.45	12.20	18.40	4.25	2.79
5.20	1.70	7.25	14.15	5.61	1.83

In calculating the columns headed "per cent. of metallic portion," it was assumed that 100 per cent. less the zinc oxide was the metallic part. The error, if any, would tend to make the figures as calculated lower than the truth. The table shows that practically any combination within the limits given for lead and cadmium may be expected to occur.

Other Impurities.—Other metallic impurities are negligible. Of the non-metallic impurities, nearly all samples contain a small amount of insoluble material, generally sand or fireclay; the amount is very small, and the only effect can be to dilute the dust with a trace of inert matter. Some zinc dust also contains a small amount of carbonaceous matter, such as coal dust, very seldom more than a small fraction of 1 per cent. This is also inert. This was demonstrated by intimately mixing 0.5 g. of zinc dust with 10 per cent. of finely ground coke dust in one case, and the same amount of graphite in another trial. In neither case did the amount of silver precipitated differ materially from that obtained when using the same amount of zinc dust by itself.

Another impurity that may occur is carbon dioxide, combined with the zinc determined as oxide. If the dust has been exposed to moist air, it is reasonable to assume that some zinc carbonate would be formed. As the solution being precipitated contains lime, the carbonate would cause the precipitation of calcium carbonate on the zinc grains, preventing efficient contact with the solution. All precipitates contain some calcium carbonate, but ordinarily, if there has not been much chance for the access of air to the solution, the amount is so small as to make the effect of carbon dioxide in the zinc dust very slight.

There is one other material that is found in many lots of zinc dust that may be of some importance. This material is oil of some sort. If present, it may be detected by extraction with ether. I do not know how it is introduced into the dust. It may be deposited in the cooler part of the condenser, and come from the reducing agent used, or it may be added later to diminish the tendency toward oxidation. On dissolving oily zinc dust in dilute acid, the action is very slow, due in part to the protection by a film of oil, and also to an effect similar to flotation methods of concentration, the dust being floated in a rather compact mass. An oil-free dust can be oiled by moistening with a solution of paraffin or some heavy oil in ether, and evaporating the solvent, and then it acts like the oily dust. It is the dust containing appreciable quantities of oil that seems to be benefited by some exposure to air; possibly the formation of a film of oxide removes the oil from the surface and leaves clean metal for precipitation.

Screen Analyses of Zinc Dust

Ordinarily from less than 1 per cent. up to nearly 5 per cent. will remain on a 200-mesh screen, and from one-fourth to three-fourths of this will remain on a 100-mesh screen. The coarse particles are partly sand, bits of fireclay, and similar things; usually there are some coarse particles of zinc. The amount that passes through a 200-mesh screen does not indicate the real fineness of the zinc dust, as much depends on the average size of the through-200 portion. A further separation may be made by elutriation.⁴ Much may depend upon the composition of the various fractions.

In some cases lead determinations were run on the separate fractions. In one lot the fine portions contained progressively less lead, and in another contained more than the coarser fractions. Cadmium determinations were not made on these fractions. As a rule, the fine portions contain higher percentages of zinc oxide, and there may be enough to outweigh the effect of a high proportion of fines. In other lots, however, the zinc oxide was fairly evenly distributed through all the sizes. It may be that determinations of metallic zinc in the fractions would be of value. The difficulty in determinations of this sort is the chance for oxidation during the manipulation, especially the drying of the fractions.

Substitutes for Zinc Dust

There have been several zinc products proposed as substitutes for dust, but none of those so far suggested have been of any consequence. They have been mechanically made, of the nature of chips or cuttings;

⁴ W. J. Sharwood, *loc. cit.*

or they have been similar in character to test lead, of varying degrees of coarseness; some have been finely crystalline. All that I have seen have the same failing of coarseness. The efficiency of various substitutes ranges from 5 to 20 per cent. of the efficiency of an average dust, by the silver-precipitation method.

Conclusions

Zinc dust satisfactory for precipitation of precious metals from cyanide solutions should satisfy the following requirements: It should be fine; that is, most of it should pass through a 200-mesh screen, and the fine portion should be very much finer than the screen opening. As the surface exposed per unit weight varies inversely as the diameter of the particles, and the number of grains varies inversely as the cube of the diameter, if the average diameter of one lot of dust is one-half that of another, the finer would have twice the surface and eight times the number of particles. The dust would remain in suspension better, and, if uniformly distributed, the distance between grains would be half that of the coarser material.

The oxide should be low, but very low oxide does not necessarily indicate a good zinc. There should be a reasonable amount of lead, over 1 per cent. As high as 4 per cent. may do no harm. Cadmium is usually present, and some cadmium may take the place of part of the lead. If oil is present, it should not be in such quantity as to prevent good mixing of the dust with the solution. Precipitation with a standard solution of silver cyanide gives a rapid method for approximating the value of a particular lot of zinc dust. If the efficiency is well over 40 per cent. (100 per cent. being the theory for complete replacement of pure zinc by silver), the zinc is satisfactory. If under 30 per cent., the dust will probably give poor results.

The condition of the metallic impurities may have much influence. It is quite possible that the composition of different particles may vary according to the time in the distillation when they were condensed, or because of mixing of lots of dust of decidedly different composition. In a case of this sort, the precipitation test is the only simple method of determining what may be expected in actual work.

DISCUSSION

GILBERT RIGG, New York, N. Y. (communication to the Secretary*).—The following statement by Mr. Herz, if not properly qualified, is apt to be misleading:

"The nature of alloys of cadmium and lead with zinc has not been studied by modern metallographic methods." This statement is quite

* Received Dec. 29, 1915.

true so far as the ternary system Pb-Cd-Zn is concerned, but the binary systems Pb-Cd, Zn-Cd, and Pb-Zn have been studied by several investigators. Furthermore, sufficient metallographic work has been done on spelter itself to give us a very fair knowledge of the high zinc corner of the Zn-Cd-Pb diagram. A list of references is given at the end of this discussion.

Mr. Herz continues: "It is known that lead is slightly soluble in zinc, and on cooling, about 1.3 per cent. of lead is left in the zinc as a solid solution, any more lead segregating as a solution of zinc in lead. Therefore, if cadmium is absent the metallic portion of the zinc may contain 1.3 per cent. of lead solution, possibly in molecular form, while any excess must be in larger particles. This can be shown by preparing an alloy and cooling it slowly, cutting out all parts showing segregations of lead, and analyzing the remainder."

Now it is quite true that in a bath of lead and zinc kept only slightly above the melting point of the latter, the upper (zinc) layer will contain approximately the amount of lead stated by Mr. Herz (about 1 per cent.). As the temperature of the bath increases, the upper layer—the solution of lead in zinc—will become increasingly richer in lead as Spring and Romanoff have shown. With falling temperature, however, the solubility of lead in zinc rapidly diminishes. Arnemann, and Heycock and Neville have placed the limit of solubility of lead in zinc at a temperature of 418°C. at 0.5 per cent. lead. At this temperature, of course, the mixture begins to freeze. Doubtless the reason that we are unable to reduce the lead content of spelter below about 1 per cent. by liquation methods is due to the fact that at the freezing temperature the metal is somewhat viscous, and holds the lead which has been thrown out of solution (above about 0.5 per cent.) in the form of an emulsion—a suspension of minute globules. Microscopic examination of spelters confirms this. Spelters containing from 1 to 1.5 per cent. lead under higher powers of the microscope show the lead as distinct globules. If Mr. Herz will prepare the "unsegregated" parts of his lead-zinc alloy, etch with nitric acid, and examine under the microscope, he will be quite convinced that the 1.3 per cent. of lead is not present as a solid solution.

Mr. Herz also states: "Cadmium is soluble in melted zinc in all proportions and insoluble in solid zinc, but separating only at the instant of solidification does not segregate in large masses." Such investigations as have been published are rather conflicting regarding the presence or absence of solid solutions of cadmium in zinc; some investigators claim that they are absent, while others indicate their presence to a limited extent. My microscopic examination of spelters containing cadmium has invariably shown the presence of solid solutions. On the other hand, with very slow cooling the cadmium separates as a definite constituent. This, of course, only applies to cadmium less than 1 per cent. In other

words, in the stable system, produced by slow cooling, solid solutions are absent; but in the metastable system, which is practically the universal case with spelters containing cadmium, solid solutions are present due to the rapid cooling.

Mr. Herz's statement: "The probability is that cadmium would increase the solubility of lead in solid as well as melted zinc," possibly is true. Under the microscope, lead is not as easily discerned in spelters containing considerable amounts of cadmium as in those free from it.

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¹ The references under the heading Cd-Pb are incomplete. The article referred to, however, gives full data on the system.

Mill and Cyanide Plant of Chiksan Mines, Korea

BY CHARLES W. DE WITT, SEIKWAN, KOREA

(San Francisco Meeting, September, 1915)

THE ore treated at the reduction plant (called Yangdei) of the Chiksan Mining Co., Korea, is brought from four of the company mines, and from the small tribute mines. The largest shipments come from Sajunkohl mine, situated about a mile and a half from the mill (Fig. 1). The ore is quartz of medium hardness. The average assay value in gold is \$6.95



FIG. 1.—SAJUNKOHL MINE OF THE CHIKSAN MINING CO., KOREA.

per ton. At the mine the ore is put into bins, and is then trammed over an 8 per cent. grade to the tipple bins above the mill, where the ore is drawn through the mill holes as desired. The capacity of the tipple bins is about 3,500 tons.

Ore from Moonsukohl, Chung Tarrie, Yang Chun, and the tribute mines is brought in by bulls and bull wagons. The bulls carry from 350 to 800 lb.; the average wagon load is about 1,200 lb. (Fig. 2). One shovelful of ore out of every ten is taken as a sample. Considerable ore is brought in from the outside mines, and this method of sampling is

used to avoid mixing the lots. However, a series of bins will be erected in the near future and an automatic sampling device installed so that the ore can be handled to better advantage—and a much safer sample secured, with less room required.

In January, 1907, ground was broken for the first two Nissen stamps, the first ore was treated in February, 1908, and the mill was in continuous operation until August of that year. The cleanup for this run was \$6,024. Owing to shortage of supplies and mill trouble the plant was closed until May, 1909, but since that time has been in continuous operation.

The output from February, 1908, to Oct. 31, 1909, was \$11,113. For the six months ended Oct. 31, 1909, 2,840 tons of ore were milled, in 159



FIG. 2.—SORTING ORE AND LOADING BULLS AT ONE OF THE CHIKSAN MINES.

milling days (this includes 11 days that two extra Nissen stamps were running). The recovery from bullion for this term was \$5,090.

In June, 1909, two more Nissen stamps were added to the mill. The mill then ran until July, 1911, when it was found advisable to increase the crushing capacity, so two batteries of five 1,250-lb. stamps each were added.

In April, 1912, one more battery of five stamps was added, and in June of the same year four batteries of the same type were added. This made a total of 39 stamps, but owing to the wear and tear on the Nissen stamps it was found necessary to replace them. In January, 1915, the Nissen stamps were taken out and are being replaced by two 5-stamp batteries. Since 1907 our plant has been enlarged from a two-stamp to a 45-stamp mill (Figs. 3 and 4).



FIG. 3.—ABOVE THE YANGDEI MILL. TRIBUTE ORE IS SHOWN ON DUMP.



FIG. 4.—MILL AND CYANIDE PLANT OF THE CHIKSAN CO.

The course of the ore through the mill is shown in the flow sheet, Fig. 5. From the tippie bins the ore is drawn through chutes into $1\frac{1}{4}$ -ton cars and trammed to the mill, where it is dumped over grizzlies, $3\frac{1}{4}$ ft. wide by 8 ft. long, the bars being set $1\frac{1}{2}$ in. apart. The oversize goes to three Blake crushers of the following sizes: 9 by 9, 10 by 14, and

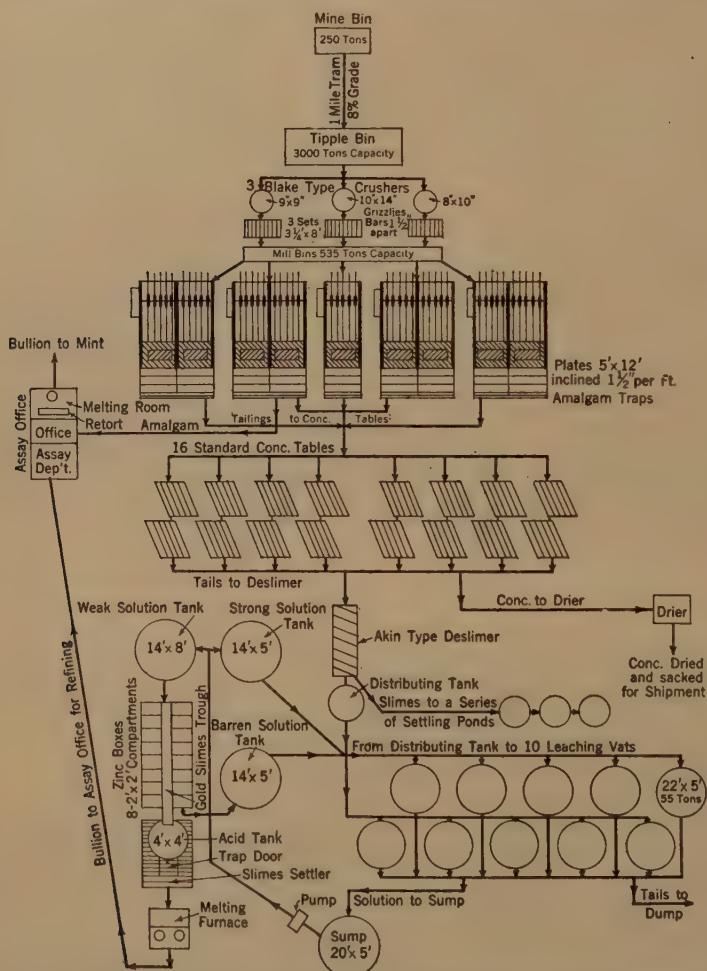


FIG. 5.—FLOW SHEET OF THE YANGDEI REDUCTION PLANT OF THE CHIKSAN CO.

8 by 10. After crushing, the ore drops direct to the mill bins. The ore then gravitates through nine Challenge feeders, which supply the nine batteries. The stamps weigh 1,250 lb. and are operated at 100 7-in. drops per minute. The distributed weight of the stamp is: Stem, 610 lb.; tappet, 150 lb.; boss head, 300 lb.; and shoe, 190 lb.

The ore is crushed to 30 mesh, and is then run over amalgamating

plates, both inside and outside amalgamation being in use. The plates are 5 by 12 ft. and have a slope of $1\frac{1}{2}$ in. to the foot. The pulp passes from the plates into Pierce amalgamating traps, and from the traps is distributed to the first series of eight concentrating tables. The middlings from these tables go to another series of eight tables on the floor below. The concentrates are dried, sacked, and shipped to the United States.

After the completion of the deslimmer, the tailings will be run direct to vats in the cyanide plant; at present they are stacked before cyanide treatment.

Cyanide Plant

In August, 1913, the cyanide plant commenced operations. At that time six vats, 22 ft. in diameter by 5 ft. deep, with a capacity of about 55 tons of damp sands, were used for leaching; during the year it was necessary to add four more vats of the same type to handle the mill tailing.

An Akins deslimmer is being installed and will be ready for operation early in 1915. The sand will then be run from the deslimmer through a rotating launder distributor, which will supply the vats; slime from the deslimmer will be run into a series of settling ponds, and if found to contain sufficient value, will be cyanided after a large enough amount has accumulated.

At present the vats are filled and emptied by coolies, carrying "jiggies." This is a very uncertain method, for when the planting and harvesting seasons come the coolies go to work on the farms, leaving the cyanide plant short handed. For this reason the plant can be run to its full capacity only a few months of the year.

The sand in the leaching vat is first treated with a 0.2 per cent. solution of potassium cyanide, supplied to the vat through a $2\frac{1}{2}$ -in. pipe from the solution tank. After about 7 hr. the solution is drained off, and while draining a 0.1 per cent. cyanide solution is run in; this is drained after a few hours, and at least two washings of water applied. The vats are then unloaded and prepared for a new charge, four days being taken for total treatment, including the loading and unloading of the vats.

The pregnant solution is pumped up to the receiving tank, from which it flows to two 8-compartment zinc boxes, having a total length of 20 ft., with compartments 2 ft. square. Zinc shavings are used as the precipitant. The solution from the boxes is run into the barren-solution tank, and is then pumped up to the strong-solution tank, where it is made up to the required strength.

The gold slime is cleaned from the boxes and run into the acid tank, where for three days it is treated with sulphuric acid. The slime is then run into a vacuum settler and drained; after which it is dried over a large iron drying pan, weighed, and melted, the resulting button being sent to the assay office for refining and assaying.

The performance of the mill for the year ended Dec. 31, 1914, was:

Tons crushed.....	46,402
Average assay value.....	\$6.956
Gross value (head assays).....	\$322,770

Bullion (21,872 oz.):

Gold.....	\$231,932
Silver.....	5,794
Concentrates (414.33 tons).....	31,388

Total.....	\$269,114
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Milling cost, per ton crushed.....	\$0.67
Average assay value of concentrates.....	\$75.75
The stamp days.....	11,190.24
Stamp duty per 24 hr.....	4.12 tons.

The costs and recovery at the cyanide plant for the year Jan. 1, 1914 to Jan. 1, 1915, were:

	Per Ton	Total
Recovery.....	\$0.799	\$29,426
Cyaniding costs.....	0.335	12,462
Profit.....	\$0.464	\$16,964

Total recovery from reduction plant was:

Bullion (28,105.6 oz.):	
Gold.....	\$259,013
Silver.....	8,062
Concentrate.....	31,388
Total output.....	\$298,463

Water for the mill is brought from the creek through a small ditch about a mile long. During the summer or dry season the water supply was insufficient to run the mill, so it was necessary to make settling ponds; the water from these ponds is pumped back to the mill.

The mill is operated by a 300-hp. Russell compound engine, which also drives a 75-kw. alternating-current generator, which in turn drives the other small 5- and 10-hp. motors and supplies the lighting service for the mine and camp. Power for the engine is supplied by three 150-hp. boilers.

Amalgamation Tests

BY W. J. SHARWOOD, LEAD, S. D.

(San Francisco Meeting, September, 1915)

THE assay or estimation of the total gold content of an ore presents little difficulty, when circumstances permit of securing a thoroughly representative sample. The general practice has been fairly standardized, and such errors as occur in the results obtained are for the most part attributable to the methods of sampling rather than to those of assaying, while duplicate assays made by skilled workers usually agree closely.

When it is a question of estimating the "free" or amalgamable gold of an ore, as opposed to the total gold, there is but little uniformity in the methods employed by different workers, and often equally little in the results obtained when duplicate determinations are attempted, no satisfactory standard method having been generally adopted. The methods in use are of two general types—"panning" and "amalgamation."

Panning

For the estimation of the value of the visible gold by panning or washing, a weighed sample may be taken, ranging from an ounce to several pounds; a kilogram or 2 lb. is a favorite quantity for laboratory tests, while a prospector often uses a full miners' pan, say nearly 20 lb. or 10 kg. In the case of rich quartz, ground to about 30 or 40 mesh, a Mexican horn is convenient, using 1 to 8 oz. of material, and a 6-in. frying pan is favored by many prospectors in dry districts. Bateas, plaques, and dishes of various patterns are used for the same purpose. With these, fairly close results may be obtained by using for ocular comparison standards consisting of weighed amounts of gold of similar character as regards purity, fineness of division, and the general shape of the particles, which standards are preferably obtained from gold of the same locality, or the same deposit, as the ore or gravel being tested. Men working in one particular district sometimes become surprisingly expert in thus judging the value of an ore by the eye, the weights of both ore and gold being guessed. On the other hand, many have been led to overestimate the value of ore and gravel from unfamiliar localities, where the gold has been very finely divided or of unusually high silver content, and a novice may

be easily misled by the presence of such minerals as lead chromate or molybdate. The presence of "sulphurets" is also often a source of difficulty.

When the visible gold obtained by panning is collected and weighed the results are generally low; when the collected pannings are fused and assayed the results may be high as regards "free" gold, owing to the inclusion of a certain amount of concentrate. Sometimes, after concentration by panning, the concentrate is treated with a little mercury or sodium amalgam, which leads to a combination process.

Amalgamation

The amalgamation assay consists in subjecting a weighed sample of ore to the action of mercury, and separating the latter after the gold is presumed to have become amalgamated. Preferably the original ore or "heads," the "tails" after amalgamation, and the mercury which has been separated, are all three assayed, as the results then check one another. It is common, however, to determine only two of these factors, usually the original ore and either the amalgamated gold or the tailing. For some reasons, especially in the case of an ore carrying coarse gold, it is preferable to assay the tailing and determine the amount of gold amalgamated, assuming the original value as the sum of these.

Inasmuch as some free gold is quite invisible, and even coarse gold may be very difficult to detect in the presence of a large proportion of sulphurets or heavy pyritic minerals, the amalgamation method has obvious advantages, especially as the results of panning depend so largely on the operator's judgment and skill in manipulation.

Numerous methods have been published for carrying out the amalgamation assay, but many of the instructions lack precision, leaving so much to the worker's judgment that they are of little use to the inexperienced, and there is a bewildering variety in the quantities of material and times of treatment prescribed. Such directions as "stir for a few hours, or until completely amalgamated," are not infrequent.

Some of the authors recommend agitation, others simple rotation in a bottle; some add water to make a thin pulp, while others specify that it must be thick enough to keep the mercury from settling. Some grind the ore with mercury in an iron mortar till it is reduced to an impalpable powder, others merely rub or stir with a wooden pestle, others again insist on a Wedgwood mortar; one uses as much mercury as ore, or even more, while another recommends less than 1 per cent. A few prescribe grinding with a large proportion of sodium amalgam, endeavoring to secure a maximum amalgamation; most, however, aim to obtain results which shall compare fairly with those realizable in a well-equipped gold mill; some try to imitate pan amalgamation by adding various chemicals

while grinding in an iron mortar. A few recommend preliminary concentration, amalgamating the concentrate, or at least begin by washing off the slime and lighter minerals; others, also using amounts too large for a single operation, divide them, and use the same mercury on several successive charges of a given ore.

Two pieces of special apparatus may be mentioned. Amalgamated copper pans, or copper-bottomed pans with iron rims, are often recommended for use when the tailing and not the amalgam is to be assayed. Their efficiency is increased by silver-plating, and sometimes by adding a rib or riffle on one side to assist in retaining the mercury. They may be made to fairly imitate the conditions of plate amalgamation. Mechanical bateas or pans suspended by springs are sometimes used for the same purpose. The iron arrastre or Buck's mortar is cast with an annular trough in which a muller is revolved by hand; in the best forms the muller is divided into several segments; when used with a fairly large proportion of quicksilver it represents closely the action of an amalgamating pan.

In the main there are two principal schools: One relies on the assay of the tailing and pays comparatively little attention to the quality or quantity of mercury used, but removes it carefully by panning; the other depends on the precise determination of the gold recovered as amalgam, and hence demands the use of mercury which is either free from gold, or contains only a small and known amount.

In determining the gold present in the mercury after its contact with the ore, very great differences also occur in the published directions; these are discussed to some extent under the head of determination of gold and silver in amalgam and mercury.

In the accompanying table an attempt has been made to compare the more important details of procedure recommended by various writers. Some of the typical methods are further summarized. Some of the variations noted undoubtedly arise from differences in the objects aimed at; some of the writers have evidently had in mind a process which could be carried out in the field by any prospector, and most of them appear to contemplate only a few occasional determinations, as the methods usually demand too much personal attention to be of every-day application.

Among all the published descriptions found, apparently only one—that of Guess—refers to the amalgamation assay as a real means of plant control; in this case it was used as a regular working guide to mill practice and as the basis for settlements in treating custom ore. Another paper—that of Warwick—deserves special mention, as, in addition to describing amalgamation tests, it treats with considerable detail the examination of an ore to decide the various forms in which the gold may occur.

Amalgamation Tests for Free Gold in Ore

Authority	Date	Method or Apparatus	Weight of Ore, Grams	Mesh	Weight of Mercury, Grams	Time of Treatment	Extraction Based on Assay of
H. Louis ¹	1893	Ground in mortar.	900 to 1,350	60	30	1 hr. ?	Mercury (cupelled).
R. W. Leonard ²	1895	Rubbed in mortar.	300 to 3,000	40 to 80	Small amount.	1 hr.	Mercury retorted.
T. K. Merritt ³	1896	Rubbed in miners' pan.	900	60 to 80	30	1 hr. to 4 hr.	Mercury (cupelled).
T. K. Rose ⁴	1896	Rubbed in mortar.	40*	50 to 75	1½ hr.	Mercury (scorified).
A. S. Warwick ⁵	1896	Stirred in mortar.	120	Each 10	1½ hr.	Tailing.
H. S. Guess ⁶	1898	Bottle agitator.	10 lots of 100	50 to 100	1 hr.	Plate and mercury parted.
L. Janin, Jr. ⁷	1903	Bottle agitator.	30 to 120	?	½ hr. for each passage.	Mercury (scorified or blow-piped).
E. A. Hersam ⁸	1904	Small copper plate; passed repeatedly.	50	Mercury (parted).
E. A. Hersam ⁸	1904	Glass U-tube and air lift.	100	Globule, 0.5.	Mercury (fused in crucible).
W. E. Darrow ⁹	1907	Bottle agitator.	900	60 or 70	6 to 7	1 hr.	Preferably tailing.
A. T. Roos ¹⁰	1910	Rubbed in mortar.	175 to 350	15	½ hr.	Mercury (fused in crucible).
J. A. Barr ¹¹	1910	Rotated in bottle.	150	200	2 to 3 hr.	Mercury (fused in crucible).
J. E. Clennell ¹²	1910	Rolls in bottle.	1,000	100*	20 to 30	1 to 2 hr.	Mercury (fused in crucible).
W. J. Sharwood ¹³	1911	Bottle agitator.	100	10	1½ to 2 hr.	Mercury (fused in crucible).
A. McA. Johnston ¹⁴	1911	Agitated in mortar (or bottle).	300	Mercury (fused in crucible).
A. McA. Johnston ¹⁴	1911	Copper pan.	500 to 1,000	80 to 100	15 to 20	1 hr.	Mercury (fused in crucible).
E. A. Smith ¹⁵	1913	Ground in mortar.	Mercury (fused in crucible).
H. Cramer ¹⁶	1731	Concentrated and ground.	140 to 200	15	Mercury (distilled).
E. W. Buskett ¹⁷	Ground in iron mortar.	150	Spoonful.	Several hours.	Mercury.
P. Ricketts and E. H. Miller ¹⁸	Ground in mortar.	180 to 240	80	Few ounces.	2 to 3 hr.	Mercury and tailing.
H. Van F. Furman ¹⁹	Concentrated and shaken hot in bottle.	450 to 1,350	80	Several hours.	Strains mercury, heats and cupsels.
H. Van F. Furman ¹⁹	Copper pan.	Few ounces.	Fine.	30	1 to 3 hr.	Tailing.
R. W. Lodge ²⁰	Shaken in pan.	300 to 500	Some time.	Tailing and mercury (parted).
R. W. Lodge ²⁰	Shaken vertically in bottle.	300 to 1,000	½ hr.
R. W. Lodge ²⁰	Revolved in bottle.	3 hr.
R. W. Lodge ²⁰	Ball mill.	2,000 to 5,000	200

* Or to correspond with milling practice.

¹ *Stamp Milling of Gold Ores.*

² *Trans.*, xxv, 645 (1895).

³ *Trans.*, xxvi, 187 (1896).

⁴ *Trans.*, xxvi, 187 (1896).

⁵ *Journal of the Society of Chemical Industry*, vol. xv, No. 3, p. 182 (Mar. 31, 1896).

⁶ *Trans.*, xxv, 187 (1896).

⁷ *Trans.*, xxv, 187 (1896).

⁸ *Trans.*, xxv, 187 (1896).

⁹ *Trans.*, xxv, 187 (1896).

¹⁰ *Trans.*, xxv, 187 (1896).

¹¹ *Trans.*, xxv, 187 (1896).

¹² *Trans.*, xxv, 187 (1896).

¹³ *Trans.*, xxv, 187 (1896).

¹⁴ *Trans.*, xxv, 187 (1896).

¹⁵ *Trans.*, xxv, 187 (1896).

¹⁰ *Mining World* (Chicago), vol. xxii, No. 6, p. 319 (Feb. 5, 1910).

¹¹ *Testing for Metallurgical Processes.*

¹² *Cyanide Handbook* (1910).

¹³ *Fulton's Manual of Fire Assaying*, 2d ed. (1911).

¹⁴ *Trans.*, xxv, 187 (1896).

¹⁵ *Sampling and Assay of the Precious Metals* (1913).

¹⁶ *Elements of the Art of Assaying Metals* (Mortimer's translation, 1764).

¹⁷ *Notes on Assaying*.

¹⁸ *Manual of Practical Assaying* (1893).

¹⁹ *Notes on Assaying* (1904).

²⁰ *Notes on Assaying* (1904).

E. A. HERSAM suggests two methods:

1. A very small piece of amalgamated copper foil is repeatedly subjected to the action of a stream of ore particles passed over it under the surface of water in a flask, the whole sample being passed over it a definite number of times, each passage taking half an hour. The copper plate is then treated with nitric acid and the gold recovered.

2. A glass apparatus with an air-lift device causes the ore, mixed with water to a thin pulp, to circulate over a globule of mercury in a narrow U-tube. The mercury is finally volatilized; the residue is fused before the blowpipe, or may be scorified, cupelled, and parted.

R. W. LEONARD divides a large sample into several lots which are rubbed successively in a mortar, the mercury separated from the first being used again for the second, and so on. Finally it is cleaned and put into a small iron retort and heated to expel the mercury. The gold is recovered by fusing a little test lead in the retort; the lead is then cupelled, "preferably with the blowpipe."

W. H. MERRITT uses about 2 lb. (900 g.) of ore crushed to 40 or 60 mesh, and rubs this with an ounce (30 g.) of mercury in a miners' pan with a wooden pestle. The mercury is "retorted" in a cup-shaped piece of sheet iron, or in a potato, and the gold weighed.

A. McARTHUR JOHNSTON.—Weigh out 10 assay tons into a Wedgwood mortar and add water to make a thin pulp. Into this place 10 g. pure redistilled mercury and agitate half an hour. No attrition should be allowed other than that between the particles themselves, and between them and the sides of the mortar. Should the ore contain any ingredient tending to foul the mercury some reagent may be added (sodium hydroxide or hydrochloric acid) to make the mercury more active. If rusty gold is present it is often expedient to add sulphuric acid. The pulp could also be agitated with mercury in a bottle, but this method causes flouring or fine subdivision of the mercury and makes its collection very difficult.

To recover the mercury, pan off the sand and concentrate, taking care not to lose minute globules; even with care a slight loss of mercury ensues. A separating funnel may also be used. The collected mercury is cleaned and weighed, and assayed for gold. The gold thus found is multiplied by the original weight, and divided by the recovered weight, to give the total gold recoverable by amalgamation; the result is calculated per ton of ore.

W. H. COGHILL also recommends the use of a small separatory funnel with two stopcocks, such as is used in mineralogical work with heavy liquids, for removing mercury from amalgamated tailing.

A. W. WARWICK places in a porcelain mortar 4 assay tons (120 g.) of ore crushed to a suitable size (or treats several such samples at different meshes), adds water enough to make a very thin paste, then 50 to 75 g. of mercury, accurately weighed. The mixture is stirred, not rubbed, with a light pestle of hard wood for $1\frac{1}{2}$ hr. It is then diluted, the thin paste poured off, and the mercury washed, dried, and weighed. The loss is indicative of the tendency to "flour" and Warwick considers amalgamation impracticable if it exceeds 1 per cent. The collected mercury is placed in a scorifier, covered with a second scorifier, placed in a muffle to vaporize, and the residue scorified with lead, cupelled, etc.

H. S. GUESS describes a method used practically for the control of a custom mill, the ore being crushed to 40 mesh, or to agree with the mill practice. From three to ten samples of 100 g. each are put in wide-mouthed stoppered bottles, with 10 g. of mercury and water, and agitated 30 min. A vertical agitator is used, having a 2-in. eccentric and run at 350 rev. per minute. He pans and discards the mercury, using a deep aluminum pan 10 in. in diameter. The concentrate is separated, dried, and assayed; the tailing and slime are dried together and assayed. The values of concentrate and total tailing are subtracted from the original assay to give the amount amal-

gamated. The returns of a number of mill cleanups are quoted to show their close agreement with these tests.

L. JANIN, JR. uses a much larger proportion of mercury than the preceding. He shakes 1 to 4 assay tons of ore with 50 to 100 g. of mercury in a bottle for an hour, by means of a "milk shake" or Gay-Lussac silver-assay apparatus. The bottle is then inverted, most of the mercury poured out, and the remaining globules separated by the use of a copper pan or dish. The proportion of free gold is determined by an assay of the original pulp and final tailing.

HERMAN CRAMER described in *Elements of the Art of Assaying* the process of amalgamating previously concentrated ores of native silver (or gold) in an iron mortar, using a wooden pestle and a large proportion of mercury, also adding vinegar and alum.²¹ The mercury is finally to be squeezed through thin leather and the solid amalgam distilled in a glass retort on a sand bath. He mentions the danger of precious metal passing into the receiver with the mercury when the fire is too strong. He adds the following caution (following Mortimer's translation): "If, for want of an apparatus for scorification and coppelling, you have a Mind to indicate by this method, the Quantity of Silver contained in the Body washed; in this case the whole Amalgama must be distilled through the Retort; because Part of the Silver and gold gets through the Leather: Nay, there remains nothing at all of the Silver or Gold within the Leather, if you use too great a Quantity of Mercury, to extract a small Quantity of these Metals: unless the Mercury be saturated with them, by a like previous Process; and even then, you may be easily deceived as to the Quantity and Quality of the Metal." . . . "Amalgamation itself is more proper to Gold than to Silver . . . A true metallic State is required for an Amalgamation, because no Extraction can otherwise be made by Mercury."

The quotations from Cramer have been introduced to show that he was alive to sources of error that have been overlooked by some much more recent chemists.

The Estimation of Gold and Silver in Mercury and Amalgam, and the Preparation of Pure Mercury

These subjects are so intimately connected that they are best treated together. It is a common delusion that only a negligible amount of gold or silver passes with the mercury through leather or cloth used in straining amalgam, and a still commoner one that mercury is completely freed from gold by distillation or "retorting."

At the ordinary room temperature mercury dissolves at least 0.03 per cent. and up to 0.05 per cent. of gold, and I have found about 0.02 per cent. gold and half as much silver in squeezed mercury which had deposited most of its contained amalgam as crystals after long standing at a temperature just below the freezing point of water. The usually quoted value of 0.03 per cent. or 300 parts per million is equivalent to \$180 per ton. The actual content is often much higher, owing either to minute particles of suspended amalgam passing through holes in the straining cloth, or to straining at a higher temperature and retention of the dissolved amalgam, apparently as a supersaturated solution.

If the attempt is made, therefore, to separate the collected gold from

²¹ This combination of vinegar and alum is also mentioned by Ercker.

the mercury in the amalgamation assay by squeezing through buckskin, it is easy to see that the results are entirely misleading. If we use only 5 g. of mercury with 100 g. of ore (or $\frac{3}{4}$ oz. per pound) at least \$9 may be extracted per ton of ore and pass through the leather with the mercury. This method is entirely inadmissible except for the roughest estimation of coarse gold.

Even when the method of assaying the tailing is adopted, the use of squeezed mercury may lead to serious error; the retention of $\frac{1}{2}$ g. in the floured condition in 100 g. of ore making the tailing too high by at least 90c. per ton.

In retorting mill amalgam an appreciable, though small, amount of gold is carried over with the quicksilver, probably owing to spurting or violent boiling. H. Louis estimates that under the most favorable working conditions at least 5 parts of gold pass over per million of mercury condensed, so that the mercury carries at least \$3 worth of gold per ton. Often much more may be found in it. If mercury carrying \$3 per ton is used in tests, at the rate of 20 g. per 100 g. of ore, the assay of the mercury will indicate a recovery which is too high by 60c. per ton of ore. If the extraction is calculated on the assay of the tailing the error thus introduced is inappreciable unless flouring is excessive.

By a cautious slow redistillation of this nearly pure quicksilver in a clean new iron retort, with a condenser of new iron pipe, it is obtainable practically free from gold. This is the best method to adopt where practicable. Vacuum apparatus may also be used for effecting the distillation under reduced pressure at a much lower temperature than is otherwise practicable, or glass condensers may be used, but neither is of much advantage for the present purpose. Hulett has shown that, if distillation is slow enough to avoid actual boiling, even such volatile metals as zinc and cadmium are carried over in only minute amounts.

It is still easier, at mills where quicksilver is bought in large quantities and comes directly from the mines, to test several of the best looking flasks for gold, and select the best found. Sometimes scarcely a trace is present in new quicksilver; occasionally a flask is found containing a considerable proportion of gold, possibly owing to the use of old mill flasks.

For the preparation of small amounts of gold-free mercury Darrow recommends treating a quantity with an insufficiency of nitric acid, so as to leave the gold in the undissolved portion. The filtered solution of nitrate is then precipitated by means of pure copper, such as electric wire. This is only suited to small amounts. Electrolysis of the nitrate is not practicable, as even the densest graphite electrodes are rapidly disintegrated. A simple method of removing gold from mercury was described by W. Bettel,²² who puts a thin layer in a shallow dish and covers it with a 2 per cent. solution of potassium cyanide, adding a little sodium per-

²² *Mining and Scientific Press*, vol. xcvii, No. 26, p. 881, (Dec. 26, 1908).

oxide at intervals. The water requires renewal occasionally. I have found this effective but rather slow; stoneware or agateware pans may be used, but are somewhat attacked by the alkali.

In estimating the precious metal in small amounts of amalgam or in mercury, two methods naturally suggest themselves. One is to distill off the mercury, the other to remove it with nitric acid.

In practice it is found that the first invariably causes a certain loss of precious metal, while in acid parting the silver is always too low. It is recommended that the acid method be adhered to, so far as gold is concerned.

In case it is desirable to estimate silver also, it is recommended to take two portions of the material. In one the mercury is removed by nitric acid, finishing with hot and fairly concentrated acid; the remaining gold being then cautiously dried and heated to expel any residual mercury, inquarted, and parted as usual. The second portion is put into a small scorifier, covered with another inverted scorifier, and set in a comparatively cool muffle, which is then slowly heated to strong redness, taking care that there is a strong draft to carry the fumes outside.²³ The residue in the scorifier is then covered with a layer of test lead, which is scorified a few minutes and cupelled. The scorifier used as a cover must be carefully examined for traces of adhering metal, and in exact work it is preferable to also scorify a little lead in the cover and add it to the main lot. Small lots of clean amalgam may be similarly treated in a covered cupel to avoid scorifying.²⁴ From the total gold obtained by the wet method, and the ratio of gold to silver indicated by the fire assay, the amount of silver may be accurately determined; such refinements as the correction for cupel absorption being introduced if warranted.

A satisfactory method of recovering both silver and gold from amalgam or mercury is to heat it in a crucible under a heavy cover of assay flux, which is then fused and the resulting lead button cupelled. Thus A. McArthur Johnston recommends a layer of 60 g. of litharge, mixed with enough reducer to yield 25 or 30 g. of lead, and a cover of borax. Ordinary assay flux, with a little silica added, appears to work well. In any case extreme care must be taken to guard against the escape of mercury fumes into the laboratory.

²³ The vapor of mercury has so high a density that a moderate draft is insufficient to carry it off, and heating in a muffle without proper precaution may lead to serious consequences. In regard to the danger of vaporizing mercury without using a condenser, the warning given by Pettus in his translation of Ercker's work may be appropriately quoted: "Take heed lest the *smoak* or vapour go not into thy *Belly*, because it is a *poysounous* and cold *Vapour*, which lameth and killeth: for, he will find that it will there congeal and afterwards spoil his body."

²⁴ It is scarcely necessary to add that the cupellation of amalgam directly, or of gold containing more than a trace of mercury, causes the "spitting" of the cupelling lead, and the contamination of other alloys.

For rapidly obtaining the gold from a small amount of amalgam the most convenient plan is to heat it in a miniature retort, made by closing a piece of glass tubing at one end and bending it to a slightly acute angle. The retort is very gradually heated to full redness and then quenched with water.

The device of placing it in a hollowed potato, or on an iron plate covered with half a potato, and heating till the potato is charred, is fairly satisfactory if the amount of gold is not too small.

When solid amalgam is treated with nitric acid a considerable amount of mercury is retained, even after long heating with strong acid, the gold protecting the mercury as it does silver. In the case of mercury containing small proportions of dissolved gold heating with moderately strong acid leaves the gold nearly pure, so that the dried gold can be safely heated without spurting and the last trace of mercury thus removed.

Extremely sloppy or dilute amalgam is very difficult to sample, as it is impossible to secure a uniform mixture of the solid and liquid portions. In sampling such material it is advisable to strain it through cloth at a rather low temperature, and weigh both solid and liquid portions. These can be assayed separately and the original content calculated, or similar aliquots can be taken from each portion and mixed for assay.

A Suggested Standard Amalgamation Test

A satisfactory laboratory method for the amalgamation test must secure the following conditions: The individual attention necessary in the separation of mercury must be reduced to a minimum, and the entire manipulation must be as simple as possible. To this end a mechanical agitator must be used, and preferably one which allows of the bottles being introduced or removed while in motion; to avoid mechanical loss bottles with some rapid but tight closing device must be adopted. Time is saved on the whole by assaying the mercury rather than the tailing, provided sufficiently pure mercury can be prepared or obtained. Sufficient mercury must be used to allow of quick action and to avoid serious error in case a few centigrams are lost. The addition of chemicals is generally to be avoided, and sodium amalgam should be used only when absolutely necessary. The cost of materials and labor must not be excessive.

The method finally to be described was evolved after much experimentation with the above considerations in view. It was intended purely as a laboratory method, the shaker requiring power; it may, however, be used at any assay office, and at a pinch can be carried out wherever nitric acid and a blowpipe are available, as two bottles can be shaken at a time by hand. The cost for materials per assay, including gasoline or fuel oil, is well under 10c.

A convenient agitator is a stoutly built wooden box with 24 or more

compartments, each 3 in. square and 6 in. deep, driven from a shaft running about 250 rev. per minute, giving a horizontal travel of about 2 in. The entire bottom, or each section, should be covered with heavy belting or linoleum. The throw may be varied by means of a wrist-pin adjustable in several holes in a disk. The box may be supported by four vertical springs of steel, $\frac{1}{8}$ by $1\frac{1}{2}$ in., 20 to 24 in. long, bolted to it and to two sills. One of this description has been in use over five years without repairs except for the replacing of a few bolts. The connecting rod is of oak, 2 by $\frac{3}{4}$ in. Where a Frue vanner is available, or any similar mechanism making 200 to 270 horizontal oscillations per minute of about 2 in., it may be utilized for this purpose. In the case of a Frue vanner this is best done by attaching supports to the oscillating frame and carrying the bottle rack across the belt.

The average percentage amalgamated in some thousands of 100-g. tests differs very little from the average stamp-mill percentage of recovery for the same period of years. Provided the samples are ground fairly fine—at least to pass 80 mesh—the results, so far as my experience goes, have differed but little with the fineness, even if ground to 200 mesh. Presumably, with the ores thus examined, the free gold is mostly exposed by a moderate degree of crushing, occurring largely, as is the case with so many gold ores, along former fracture surfaces.

This agitator and the magnesium citrate bottles have been found exceedingly useful for such work as preliminary cyanide tests; when the capacity of the bottle is too small it is convenient to take a number of portions and either make parallel tests or mix the products for assay.

Equipment Required for Systematic Amalgamation Tests

Agitator with 24 compartments, as described above.

Twenty-four "citrate magnesia" bottles, with spring clamps and rubber washers. These have a capacity about 350 to 370 cc.

Twenty-four Griffin beakers, capacity about 250 cc.

Two enameled-iron pans, rectangular, with straight sides, to hold 12 beakers each.

Filtering rack with 12 $2\frac{3}{4}$ -in. funnels and 12 extra beakers.

Hot plate.

Glass-stoppered burette for mercury, standing over an enameled pan.

Robervahl scale, turning with $\frac{1}{2}$ g.

Weighing scoop and steep-sided copper funnel or hopper for charging ore into bottles.

Cylinder, graduated to 100, 150, and 200 cc.

Six-inch funnel for water.

A few porcelain dishes about 3 in. diameter, and some enameled pans about 8 in. diameter and about 2 in. deep.

Filter paper, 12.5 cm., S. and S. No. 595 or similar quality.

Scorifiers, 2 in. or less in diameter; these may be used repeatedly and are best glazed before first using.

Washbottle.

Test lead.

Nitric acid, diluted with about double its volume of water.

Mercury, pure or previously assayed.

Silver foil, cut in pieces of about $\frac{1}{2}$ g. each.

Routine Method for Gold Only

One hundred grams of crushed ore or tailing are weighed out, charged into a bottle by means of the metal funnel, 150 cc. of water added, then 1.5 to 2 cc. of pure mercury from a burette. The stopper is then clamped, the bottle folded in a thick piece of cloth or flannel and put into the moving "shaker" for 2 hr. It is then removed, opened, covered with the thumb, shaken and inverted over a 3-in. porcelain dish, and as much as possible of the clean mercury allowed to run out. It is then shaken again for a moment, and if necessary a little more water is added, and more mercury allowed to run out into a second dish, and so on, as long as any mercury comes out. If not floured the mercury is nearly all removed in two operations. The clean mercury thus obtained is transferred to a beaker. The bottle is again shaken well and inverted so as to let a little sand run out into the dish; this is then panned into an enamel dish. Usually only a few globules are obtained; if much is found the whole charge must be panned. If "floured" a small globule of liquid sodium amalgam should be added at this point.

To the mercury, after collecting in a 250-cc. beaker, is added $\frac{1}{2}$ g. of pure silver foil. If this has been carefully prepared and cleaned by treating with cyanide solution or weak nitric acid, or by slightly amalgamating the surface, it may be used to pick up the small globules of mercury obtained in panning. Nitric acid (about 150 cc. of sp. gr. 1.14 or 1.15) previously warmed to about 70° C. is now poured into the beaker; all the beakers are set together in a pan, which is placed on the hot plate and left until all the mercury has dissolved. If not too hot it is unnecessary to use covers. As soon as the mercury disappears the liquid is filtered, the residue is rinsed on to the paper, washed once or twice with very dilute nitric acid (not over 5 per cent.) and once with water. The paper is then sprinkled with test lead, folded, and placed in a small glazed scorifier, and covered with lead enough to make up the total to 15 or 20 g. Enough pure silver is added to insure ready parting, and 2 or 3 g. of borax glass. The scorifiers are then charged into a muffle, the paper carefully burned, the lead scorified a few minutes, poured, and cupelled. The cupel buttons are parted in porcelain cups, and the gold annealed and weighed.

A correction is subtracted for gold contained in the silver and mercury.

Using a 100-g. sample, each milligram of gold represents 0.29166 oz., or \$6.03 amalgamable gold per ton.

Notes

If preferred 4 assay tons may be used, when each milligram indicates exactly 0.25 troy ounce; or 120.5 g., when each milligram indicates exactly \$5 per ton. With these amounts the water must be proportionally increased, and the manipulation in the bottles is not quite as easy as with 100 g.

In ordinary routine work the tailings are discarded. For special purposes, as when they are to be tested by cyaniding, concentration, etc., or if the original assays are unsatisfactory by reason of the presence of coarse gold, they are carefully panned free of mercury and caught in one of the larger pans, allowed to settle completely, decanted, and if necessary dried for further tests or for assay.

In special work, and particularly in case there is a tendency to flouring, the mercury is weighed out to the nearest centigram (any sodium amalgam being similarly weighed and added to it), and the recovered mercury is also weighed after drying. The recovered gold, multiplied by the former weight, and divided by the latter, gives the total gold amalgamated. In ordinary work with fine-ground ore the loss of mercury runs between 1 and 2 per cent.; and is less if the entire amount of tailing is panned.

The ore should be ground to a fairly constant degree of fineness. The product of a disk grinder, set for about 100 mesh for use in assay work, has been found to give sufficiently uniform results for most purposes. For careful comparative tests it would be preferable to sift to a definite size.

Using 100 g., 150 cc. water is preferable to 200 cc.; with less water the pulp is too thick. Material of unusually high or low specific gravity may require special attention as regards thickness of pulp. Reducing the time from 2 hr. to 1½ hr. *generally* gives good results; a less time is not safe; 1¾ to 2 hr. appear to be safe limits.

The temperature should be nearly constant from day to day; if much above the normal the results are somewhat higher; if it falls nearly to the freezing point they may be several per cent. low.

The concentration of acid recommended (one volume of concentrated nitric acid to two of water, giving about sp. gr. 1.14) was found advantageous after a number of tests. It allows of warming in advance so as to expedite action, without becoming too violent at any stage. The amount recommended leaves a sufficient excess to prevent separation of mercury salts. If concentrated during the heating it may be necessary to dilute slightly before filtering.

Addition of silver to the mercury reduces the time required for solution by about one-half, as determined by a large number of comparative tests. Commercial proof silver invariably contains gold; samples examined have carried about 0.01 mg. per half gram; or about the same amount as found in 20 g. of some of the best samples of mercury tested.

The correction for gold in mercury and silver is best determined by taking five times the amount to be used per assay, parting them together and carrying out the whole procedure exactly as in the regular work, but using five times the standard amount of test lead, scorifying off the greater portion. To avoid repeating this blank test unnecessarily it is well to have a considerable reserve stock of both mercury and silver. A thousand determinations using 1.5 to 2 cc. (20 to 27 g.) require 45 to 60 lb. of mercury, and about 16 oz. of silver.

With ores carrying less than \$3 gold per ton it is advisable to weigh out two or more parallel lots and combine the gold after parting.

If amalgamable silver is to be determined in the ore it is best to make duplicate tests, parting one and scorifying the other, as indicated on page 160.

Recovery of Mercury from Amalgamation Tailing, Buffalo Mines, Cobalt

E. B. THORNHILL,* E. M., COBALT, ONT., CANADA,

(San Francisco Meeting, September, 1915)

IN this paper on the recovery of mercury as sulphide, from the residues from the amalgamation and cyanide treatment of high-grade ores and concentrates, I will not discuss the many reactions, chemical and other-

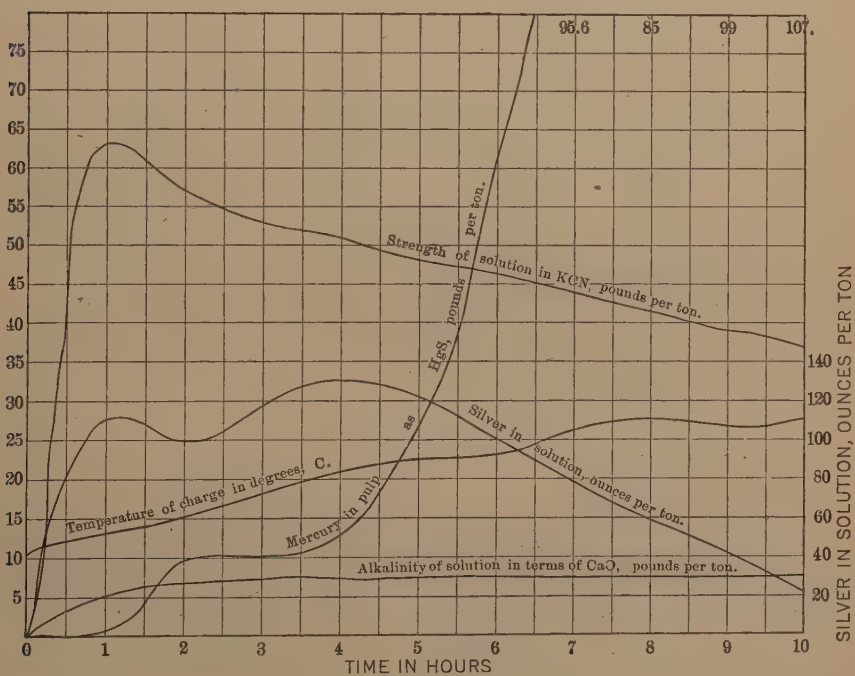


FIG. 1.

wise, that take place in the general process, but confine myself more particularly to the methods of recovering the mercury.

I submit charts of two amalgamation-barrel charges, Figs. 1 and 2, which are self-explanatory. The flow sheet of the high-grade and the mercury extraction plants is shown in Fig. 3.

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In the amalgamation of high-grade silver ores and concentrates in strong cyanide solution, as practiced in the Cobalt district, considerable mercury is retained in the residues from the amalgamation process. The greater part of this mercury is in the form of mercuric sulphide, from 5 to 10 per cent. only of the total mercury content being in the metallic state.

Attempts to eliminate this loss in the amalgamation process were made, but all resulted in a low extraction of the silver in the ore, and attention was then directed to the recovery of the mercury from the residues.

The process developed at the Buffalo Mines, T. R. Jones, General Man-

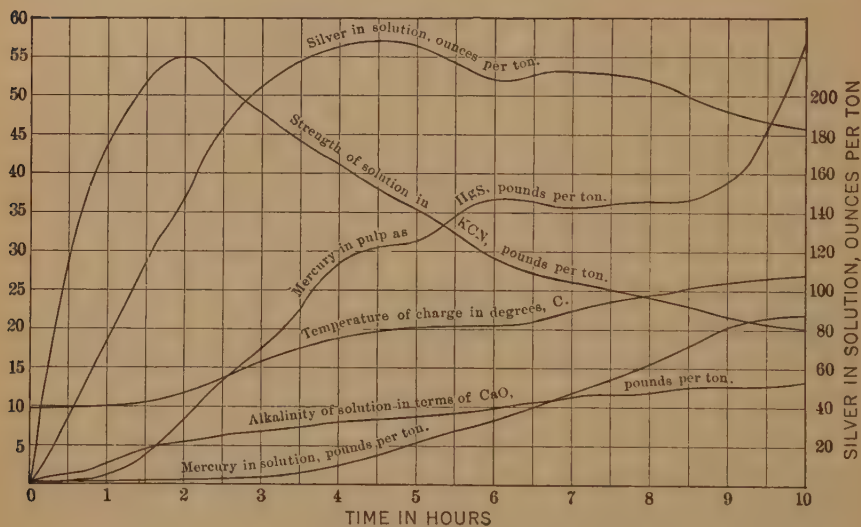
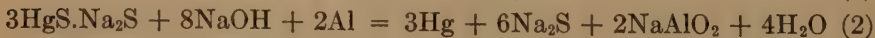


FIG. 2.

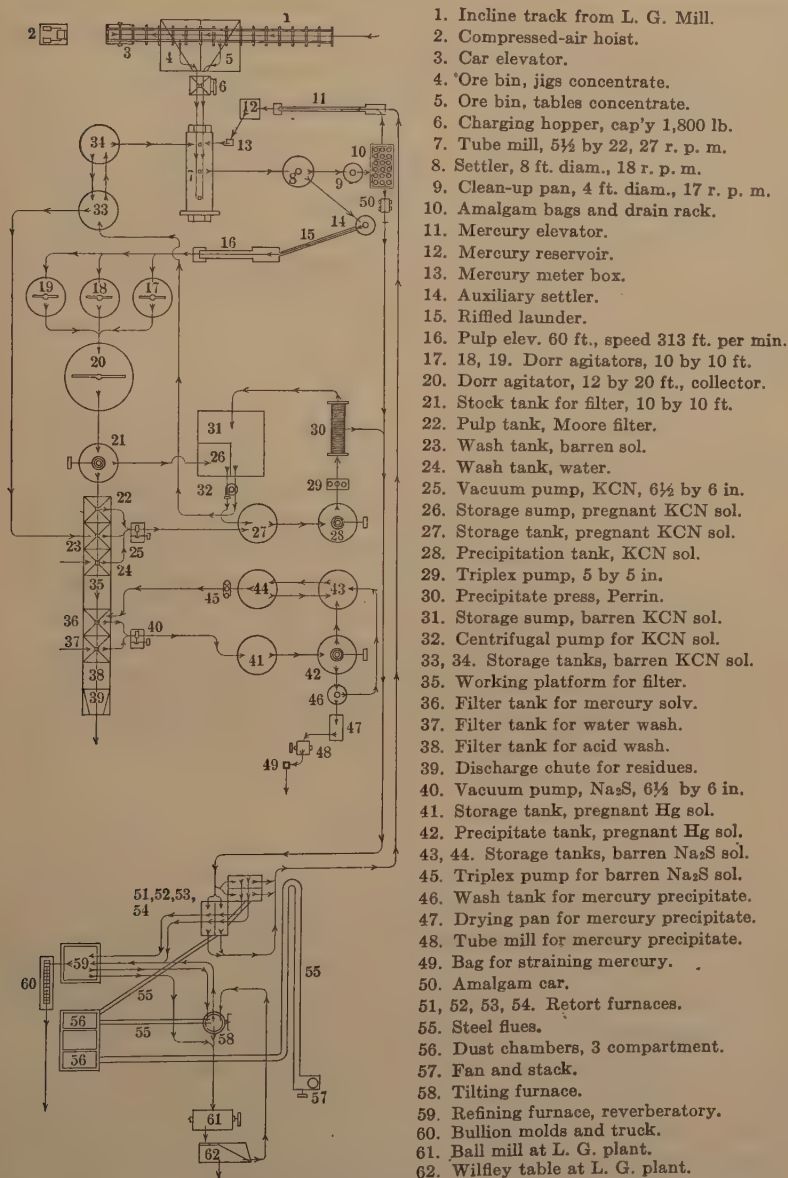
ager, for this purpose consists briefly in leaching out the mercuric sulphide with a caustic, alkaline sulphide solution, then precipitating the mercury from solution with metallic aluminum. The equations for solution (1) and precipitation (2) are:



Small-scale experiments showed that a complete extraction of the mercuric sulphide could be made by an 8 to 10 min. treatment of the residues with the alkaline sulphide solution. Advantage was taken of this fact in the commercial plant, by applying the solvent to the residue on the filter leaf, as no agitation of any sort was required.

The operation of the commercial plant is essentially as follows: The residue in the pregnant cyanide solution is caked on a Moore filter

leaf of the usual construction and the cake washed free of silver solution with water. The basket is then lowered into the sodium sulphide



1. Incline track from L. G. Mill.
2. Compressed-air hoist.
3. Car elevator.
4. Ore bin, jigs concentrate.
5. Ore bin, tables concentrate.
6. Charging hopper, cap'y 1,800 lb.
7. Tube mill, 5½ by 22, 27 r. p. m.
8. Settler, 8 ft. diam., 18 r. p. m.
9. Clean-up pan, 4 ft. diam., 17 r. p. m.
10. Amalgam bags and drain rack.
11. Mercury elevator.
12. Mercury reservoir.
13. Mercury meter box.
14. Auxiliary settler.
15. Riffled launder.
16. Pulp elev. 60 ft., speed 313 ft. per min.
17. 18, 19. Dorr agitators, 10 by 10 ft.
20. Dorr agitator, 12 by 20 ft., collector.
21. Stock tank for filter, 10 by 10 ft.
22. Pulp tank, Moore filter.
23. Wash tank, barren sol.
24. Wash tank, water.
25. Vacuum pump, KCN, 6½ by 6 in.
26. Storage sump, pregnant KCN sol.
27. Storage tank, pregnant KCN sol.
28. Precipitation tank, KCN sol.
29. Triplex pump, 5 by 5 in.
30. Precipitate press, Perrin.
31. Storage sump, barren KCN sol.
32. Centrifugal pump for KCN sol.
- 33, 34. Storage tanks, barren KCN sol.
35. Working platform for filter.
36. Filter tank for mercury solv.
37. Filter tank for water wash.
38. Filter tank for acid wash.
39. Discharge chute for residues.
40. Vacuum pump, Na₂S, 6¼ by 6 in.
41. Storage tank, pregnant Hg sol.
42. Precipitate tank, pregnant Hg sol.
- 43, 44. Storage tanks, barren Na₂S sol.
45. Triplex pump for barren Na₂S sol.
46. Wash tank for mercury precipitate.
47. Drying pan for mercury precipitate.
48. Tube mill for mercury precipitate.
49. Bag for straining mercury.
50. Amalgam car.
- 51, 52, 53, 54. Retort furnaces.
55. Steel flues.
56. Dust chambers, 3 compartment.
57. Fan and stack.
58. Tilting furnace.
59. Refining furnace, reverberatory.
60. Bullion molds and truck.
61. Ball mill at L. G. plant.
62. Wilfley table at L. G. plant.

FIG. 3.—FLOW SHEET OF THE HIGH GRADE AND MERCURY EXTRACTION PLANT.

solution and this solution drawn through the cake until the effluent shows only a trace of mercury. Usually 1 ton of solution per ton of residue is sufficient. This mercuric sulphide solution is pumped to a

precipitating tank and the mercury thrown down by adding granular aluminum to the agitated solution. Agitation is then stopped, the precipitate allowed to settle, and the clear solution decanted. The precipitate of mercury is then run into a small wash tank by sweeping it through a hole in the bottom, by means of a raking mechanism, pieces of old rubber belting being riveted to the bottom of the rakes. The precipitate is then washed with water by decantation and drawn off into a steam drying pan. After drying, the fluid mercury and the powdered metallic portion are separated by raking the latter off with a hoe. The fluid is strained through canvas and is ready to return to the circuit. The powdered material, containing approximately 75 per cent. mercury, is then retorted, and the mercury condensed in the usual manner.

The strength of the solvent used is kept up to 4 per cent. sodium sulphide and 1 per cent. sodium hydroxide. Weaker solutions can be used with equally good results, but the quantity of solution required was found to be directly proportional to the strength; that is, with a 4 per cent. sodium sulphide solution, 1 ton of solution would extract the mercuric sulphide from 1 ton of residue, but if a 2 per cent. solution were used, 2 tons would be required per ton of residue treated. The concentrated solution offers the advantage of less solution to handle and an economy of aluminum in the precipitation. The sodium sulphide used is the commercial salt costing \$1.25 per cwt. in barrels f.o.b. cars Toronto.

The precipitant used is a waste product of aluminum casting foundries, containing 75 per cent. aluminum, the impurities consisting of varying proportions of copper, silica, wood, waste, grease, etc., the grease being burned off before using the aluminum. About $\frac{1}{4}$ lb. of this material is used per pound of mercury precipitated.

It was anticipated that the sodium aluminate would accumulate in the solution to such an extent that some special means would have to be taken to remove it. This, however, has not been the case. Some aluminum as the hydrate, falls with the mercury on precipitation, and some is removed during the process of leaching, presumably as calcium aluminate, which collects on the filter cake. This precipitation of the aluminum regenerates caustic soda so that the consumption of this chemical is reduced to $\frac{1}{10}$ lb. per pound of mercury recovered, instead of $\frac{1}{2}$ lb. as shown by the theoretical equation for precipitation. Sodium sulphide is also regenerated in precipitation, but there is also a mechanical loss of approximately 20 per cent. of the solution used in leaching, as no water is used to recover the retained sulphide.

From May, 1914, to March, 1915, 37,650 lb. of mercury have been recovered at a cost of approximately 13c. per pound for labor and chemicals.

The laboratory methods of determining the mercury in the different products of the process may be of interest. To determine the total

mercury (both metallic and sulphide) 0.5 g. of the material is well mixed with cast-iron filings, free from grease, and placed in a hard-glass tube, sealed at one end, with a contraction at about 2 in. from the sealed end. The mercury is distilled off, by heating the bulb containing the charge, and condensed in the tube just beyond the contraction. After the distillation is complete, the contracted portion of the tube is heated, the bulb portion pulled off and the end of the tube sealed. The tube containing the condensed mercury is allowed to cool, filled one-half full of 0.1 per cent. KCN solution, and 10 to 15 100-mg. gold beads added. Each bead will amalgamate with about 1 mg. of mercury. The tube is shaken until all the mercury is amalgamated; the beads are transferred to a small porcelain cup, washed with water, dried with alcohol and weighed. After retorting off the mercury, they are again weighed. From 10 to 12 determinations can be made in one hour by this method and it is accurate enough for control of operations.

When it is necessary to know the proportionate amounts of mercury as metallic and sulphide two 2-g. samples are weighed out. One sample is digested with concentrated nitric acid and the other with a 10 per cent. solution of sodium sulphide. The mercury in the residue from each is then determined by the method just given.

The strength of the sodium sulphide solution is determined by titrating against a standard zinc chloride solution, using sodium nitro prusside as an outside indicator.

The mercury produced by this process is of exceptional purity. A. R. Ledoux & Co. report that 0.25 oz. silver is the only impurity in determinable quantity.

DISCUSSION

D. B. HUNTLEY, Oakland, Cal.—About 15 years ago it chanced to be my lot to cyanide some mill tailings, assaying about \$5 per ton in gold and a few cents in silver. It was in southern Idaho, a desert region where costs are nearly as large as in central Nevada. We knew that we had a lot of tailings that had been amalgamated and contained about 1 lb. of quicksilver per ton. We knew that we would dissolve some of the quicksilver when we cyanided them with say 0.25 per cent. cyanide solution. We figured (optimistically) that the cyanide would dissolve all of the quicksilver, and we thought to capture all of that, 1 lb. per ton; and we figured out profits as partly gold, partly silver, and partly quicksilver from the recovery in working those tailings. Actually, however, we did this: We had a battery of three large, old-fashioned silver-mill retorts. We retorted our zinc-dust precipitate, recovering about $\frac{1}{4}$ lb. of quicksilver per ton treated. That was only a loss of 75 per cent. of what we originally had, but still it was profitable by virtue of the

conditions. I mention it partly to call attention to the crude old way of doing it, and partly to call attention to how chemical methods for recovering quicksilver have developed in the last 15 years.

H. G. S. ANDERSON, San Francisco, Cal.—It may be of interest to mention, in reference to what the previous speaker has said concerning his experience in southern Idaho, that we tried for a while to precipitate silver from solution with zinc shavings, but we had five compartments in each box; after 30 min. there would be no precipitation in the first compartment at all; the mercury, precipitating before the silver, would cover the shavings in the second compartment in about $\frac{3}{4}$ hr., and in from 3 to 4 hr. there would be no precipitation taking place in the entire precipitation box. Then we turned to zinc dust, and zinc dust did not give us as clean a silver bullion as aluminum did, so finally, although the cost was a little higher, aluminum was accepted as the precipitant and the bullion was in much better condition and easier to smelt than it was either in the case of zinc shavings or of zinc dust.

Electric Furnace for Gold Refining at the Alaska-Treadwell Cyanide Plant

BY W. P. LASS, *B. S., JUNEAU, ALASKA

(San Francisco Meeting, September, 1915)

THE gold precipitate from the zinc-dust presses in the cyanide plant of the Alaska-Treadwell Gold Mining Co., Treadwell, Alaska, is treated, in the refinery adjoining, by the Tavener or lead-smelting method. About 3 or 4 tons of this precipitate is produced monthly having a gold assay value of \$40,000 to \$60,000 per ton.

It was formerly the practice to treat the byproducts from this process, consisting of 2 tons of slag, 300 lb. of matte, 500 lb. of refinery refuse, flue dust, etc., in a 24-in. water-jacketed blast furnace. This practice was discontinued in the summer of 1914 by reason of the difficulty of keeping the lead-well open when treating a high-grade lead product; of preventing the loss of gold in the flue dust; and of avoiding injury to the general health of the refinery operators; a plain single-phase electric furnace was substituted for the blast furnace.

The furnace was constructed from an old steel acid drum by cutting off the top and introducing a cable, made from strands of bare copper wire, through the bottom and spreading the strands out fan-shaped on the inside of the drum.

Powdered graphite, obtained by grinding up old crucibles, mixed with 10 per cent. cement, was tamped wet into the bottom of the drum, around and completely covering the copper wires. The graphite was carried up to the bottom of the furnace, or lead-well, and acted as the lower electrode. The sides were built up of ordinary firebrick forming a melting chamber 14 in. in diameter by 20 in. high.

The upper electrodes, of graphite or carbon, are 3 in. in diameter and 40 in. long, arranged with joints enabling new electrodes to be connected without shutting down or wasting stubs. A screw feed was arranged for raising and lowering the upper electrode. (Figs. 1 and 2.)

The cover for the furnace had three openings, one for feeding the charge, one for the escape of gases, and one in the center for the introduction of the electrode. It was later found more practical to enlarge this center opening to 6 in. in diameter, to allow of a central feeding of the charge around the electrode.

* Formerly Cyanide Superintendent, Alaska-Treadwell Gold Mining Co.

A 4-in. pipe connected to a ventilating fan carries off the escaping gases.

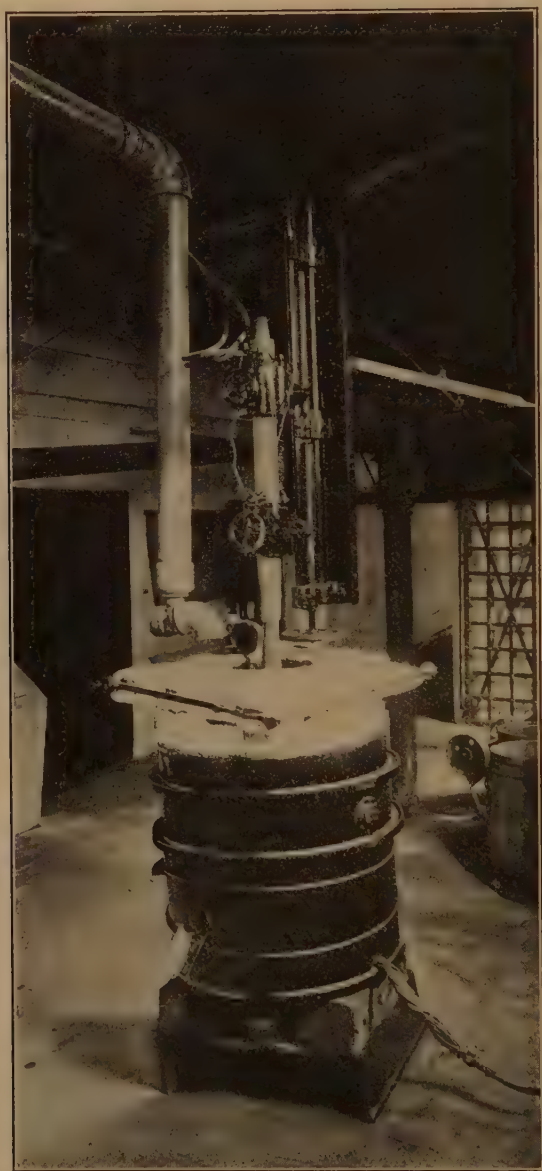


FIG. 1.—ELECTRIC FURNACE FOR GOLD REFINING.

The furnace is operated on the lighting circuit through a 50-kw. transformer, 60 cycles, 110 volts.

Later, a water rheostat, operated by raising and lowering one disk on another submerged in a barrel of water, was constructed to lower

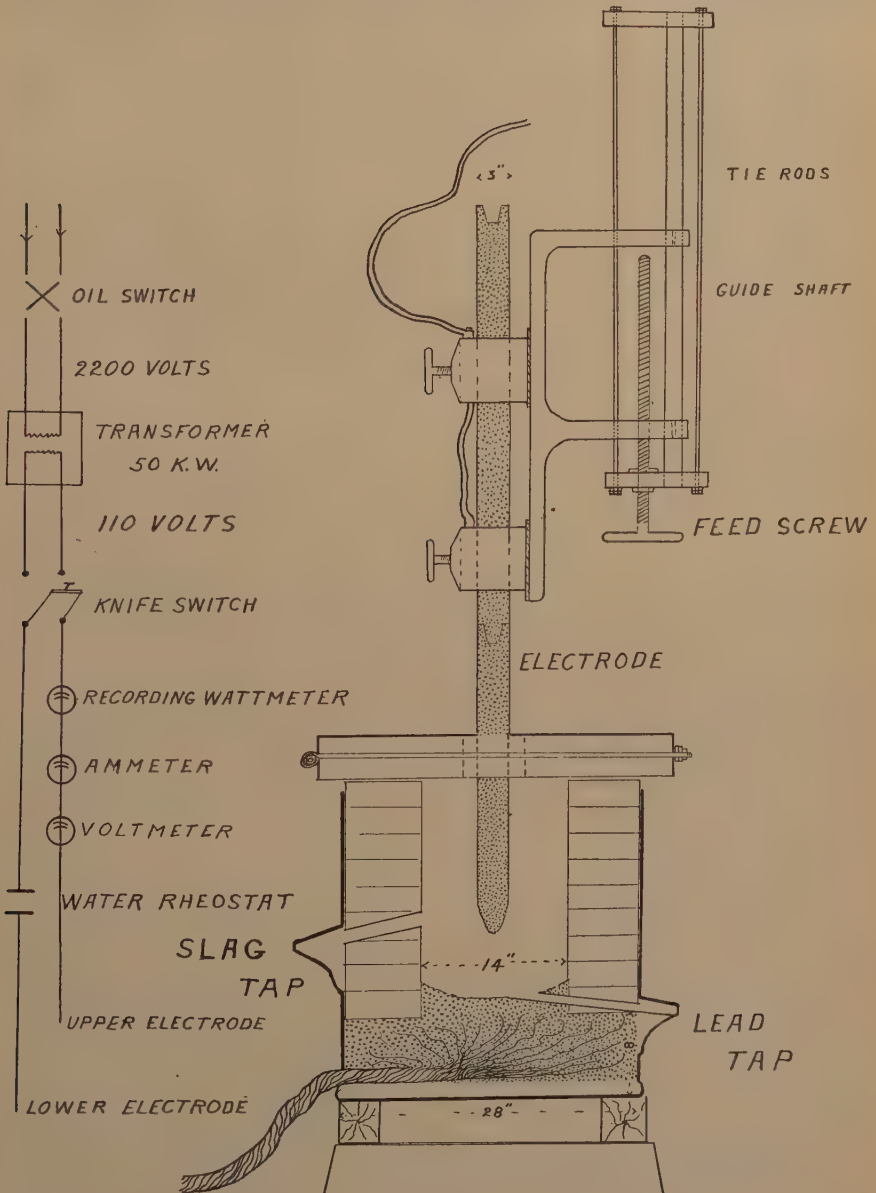


FIG. 2.—SECTIONAL VIEW OF ELECTRIC FURNACE.

the voltage for use when starting the furnace, or when there was a bath of metal in the bottom of the furnace that would otherwise allow too much current to pass and thereby cause short circuiting.

The material to be treated in the furnace is mixed in the following proportions: Refinery byproducts, 100 lb.; old reverberatory hearths, containing 60 per cent. CaO, 20 lb.; litharge, 20 lb.; coke, 2 lb.; scrap iron, 3 lb. About 160 lb. of the mixture is added for a charge. The total amount of material added during the melt is 8,507 lb., of which 5,870 lb. is byproduct material (old slag, the residue from burning old mill launders, sweepings, etc.) of the following composition:

	Per Cent.
SiO ₂	40.1
Fe.....	16.0
CaO.....	16.1
Cu.....	5.9
Pb.....	3.0
Zn.....	5.7
Al ₂ O ₃	2.0
S.....	3.6
Moisture.....	4.0
	<hr/>
	96.4
Gold.....	\$1,095.50 per ton.

Coke is added to the charge as a reducing agent, only a quantity sufficient to throw down the lead being used.

The charge is introduced into the top of the furnace through the central opening around the electrode without removing the slab cover.

In starting, the furnace is operated as an arc furnace, until it has become thoroughly hot, when the charge, consisting of slag, brick dust, etc., is added, the electrodes being raised as the load in amperes increases, until the entire chamber is filled with the charge and the upper electrode extends into the melt almost a foot. As the charge continues to melt the electrode is moved down or up, keeping the amperage reading as nearly constant as possible. The raising or lowering of the upper electrode decreases or increases the power consumption and thereby affects the amount of heat generated.

At this stage the furnace is no longer operating as a simple arc furnace, but has automatically transformed itself into a resistance type of furnace, the semi-fluid or molten charge acting as the resistant, the current passing from one electrode through the center of the charge to the other.

After the charge is in quiet fusion, which takes on an average 2 hr. 10 min., the power is turned off and the charge allowed to settle for 15 min. before tapping the slag. This allows the lead to settle out, the furnace acting as a forehearth.

In operation, the molten lead and slag are tapped, as in the blast furnace. If the lead freezes it is melted by diverting current from the

upper electrode and letting it pass through the lead, the tapping rod playing as an arc on the side of the furnace or into the lead tap.

The operators use colored or amber glasses to protect their eyes from the glare of the electric arc.

Operating Data

Total running time.....	128 hr. 40 min.
Average weight of charge added, pounds	160.5
Average fusion time of charge.....	2 hr. 25 min.
Total number of charges.....	53
Graphite electrodes used.....	1 per 24 hr.
Total power for 128 hr. 40 min., kilowatt-hours.....	4,440
Power used per hour, kilowatts.....	34.50
Power used per pound of material fed, kilowatt-hours... ..	0.52
Power used per ton of material fed, kilowatt-hours.....	1,044.00
Acheson graphite used per hour, feet.....	0.15
Material melted per hour, pounds.....	67.70
Material melted for 24 hr., pounds.....	1,625.00

The only item of cost for flux is the coke, 2 lb. per 100 lb. of by-product material or 118 lb. total being used, since the old reverberatory hearths furnish the lime, the cupels from bullion refining the litharge, and mill scrap the iron.

Both graphite and carbon electrodes are used, the former costing \$2.95 each, and the latter \$1.20. Power cost is less than 1c. per kilowatt-hour.

The labor required for the operation of the furnace, including the work of charging and tapping, is one-half one man's time, when melting 1 ton in 24 hr.

The furnace was constructed and put into operation by the regular cyanide-plant crew, without the aid of special electricians. The operation is so simple as to require no special training. Although the operators were unaccustomed to handling electrical equipment, no trouble has been experienced from electrical shocks, since the bottom of the furnace acts as the lower electrode, or grounded circuit.

The advantages of the electric furnace compared to the blast furnace for melting high-grade gold slags are: A saving in mechanical loss of gold in flue dust, because the melting is done in a quiet neutral atmosphere, instead of in a rising blast of air; the obtaining of a lower-grade slag, free from shot, by reason of the quieter melting action and the higher temperature obtainable, making a more fluid slag; the nicety of regulation of the melting temperature; the benefit to the general health of the operators.

DISCUSSION

ROBERT M. KEENEY, Somersville, Conn. (communication to the Secretary*).—There is not only a possibility of improvement in present

* Received Aug. 24, 1915.

methods of treatment of the byproducts of refining in the cyanide plant by the use of the electric furnace, but also in melting and refining of gold precipitate. This interesting paper by Mr. Lass shows an electric furnace in successful operation in a large plant for the treatment of by-products. Electric melting of cyanide precipitate is a cheaper form of treatment in many cases than any of the combustion methods now used because of the high cost of oil, coke or coal in many camps, while on the other hand electric power may be had cheaply, especially in a place where there is possibility of development of high-head hydro-electric plants for power purposes. Also, if the oil-fired tilting furnace is used for direct melting of the precipitate, as is the case at many small mills, a great saving in cost is made by the use of the electric furnace because of the high crucible cost in melting with the tilting furnace.

A study of this paper indicates the cost of the operation approximately. Consider all the fluxes to be turned in as byproducts and coke at \$18 per ton.

Cost of Electric Smelting of Byproducts at the Alaska-Treadwell

	Cost per Ton
1,044 kw.-hr. at 1c	\$10.44
Labor.....	5.25
20.4 lb. graphite electrodes at 18c	3.77
404 lb. coke.....	0.36
Total.....	\$19.82

It would have been interesting to have had figures on the gold losses. Unless there is a higher extraction in the electric furnace which would offset the cost of smelting, it is probable that in this case blast-furnace smelting of the byproduct material is the cheaper method.

Recently I made an investigation of the cost of melting precipitate from the zinc-dust presses in the electric furnace as compared with the cost of the present process used at a certain mill, melting in an oil-fired tilting furnace. The precipitate is now dried in a wood-fired muffle furnace, then mixed with the proper fluxes and melted to bullion. At periods the ore entering the mill carries considerable copper which enters the solution and comes down in the precipitate. The bullion is from 200 to 600 fine in gold and silver, of which about half is silver. The mill is 25 miles from the railroad so that the cost of fuel oil is high, 25c. per gallon, and the cost of coal and coke so high as to prohibit their use. An abundance of wood is available from which charcoal could be made, but the cost of this would be close to \$20 per ton. Hydro-electric power was developed in the district at a cost of 0.31c. per kilowatt-hour and the cost of any more power developed would not exceed 0.5c. per kilowatt-hour.

During a period when there was considerable copper in the ore, probably resulting in a somewhat higher melting cost than usual, 2,161 lb. of dried precipitate was melted, producing 7,596.25 oz. of bullion. In the following comparison of costs it is assumed that the precipitate is dried in the muffle furnace before treatment in either the oil furnace or the electric furnace.

Cost of Melting Precipitate in the Tilting Oil Furnace

		Per Pound of Precipitate
691 lb. borax at 15c.....	\$103.65	\$0.048
178 lb. soda at 11c.....	19.58	0.009
6 crucibles at \$11.50.....	69.00	0.032
318 gal. oil at 25c.....	79.50	0.037
Furnace repairs.....	8.00	0.003
Labor for melting.....	60.00	0.027
Supervision.....	25.00	0.011
Total.....	\$364.73	\$0.167

Estimated Cost of Melting Precipitate in a 50-kw. Electric Furnace

		Per Pound of Precipitate
691 lb. borax at 15c.....	\$103.65	\$0.048
178 lb. soda at 11c.....	19.58	0.009
30 lb. carbon electrodes at 10c.....	3.00	0.001
1,515 kw.-hr. at 5c.....	7.58	0.003
Furnace repairs.....	10.00	0.004
Labor.....	18.00	0.008
Supervision.....	25.00	0.011
Total.....	\$186.81	\$0.084

In making this estimate the electrode consumption is considered as 20 lb. of carbon per ton of charge, and the power consumption is taken as 1,000 kw.-hr. per ton of charge. The lower labor cost of the electric furnace is simply due to its capacity, and depends upon the power load on the furnace. An electric furnace of smaller size might melt at the same rate as the oil furnace when the labor cost would equal that of the oil furnace, but there would still be a considerable saving in other items.

These figures show a saving of 8.3c. per pound of precipitate melted in favor of the electric furnace, or a saving of 50 per cent. The main saving is in the source of heat and the cost of crucibles as compared with electrodes. Electric heating costs 0.3c. per pound of precipitate and oil heating costs 3.7c. per pound. Electrodes are 0.1c. per pound of precipitate and crucibles 3.2c. per pound.

For small mills where direct melting is the most feasible process, it appears that the electric furnace will melt more cheaply than the oil

crucible furnace, at least where the cost of electric power is not exorbitant and fuel oil is expensive. With oil at 5c. per gallon and power at 3c. per kilowatt-hour, the melting cost in the oil-fired furnace is 13.7c. per pound and in the electric furnace 10c. per pound showing that there is application for the electric furnace in regions not isolated from the source of oil.

It should be possible to keep the losses in the electric furnace as low as in a combustion furnace. The slag loss would be as low, and experience has shown that with proper manipulation it is possible to operate an electric furnace so as to keep down volatilization to a low figure. There would be no gold loss, and silver loss should not be higher than in the combustion furnace.

R. S. WILE, Pittsburgh, Pa. (communication to the Secretary*).—H. R. Conklin of the Lluvia de Oro Mining Co., Lluvia de Oro, Chihuahua, Mex., was the first man to attempt the electric smelting of gold zinc-dust precipitates, after investigating two furnaces I had in operation at Connellsville in the fall of 1910. He published an article in the *Engineering and Mining Journal*, 1912, covering the operation of this furnace. The furnace that Mr. Lass describes is the duplicate of this one, showing no points of originality.

* Received Sept. 3, 1915.

A Rule Governing Cupellation Losses

BY W. J. SHARWOOD, LEAD, S. D.

(San Francisco Meeting, September, 1915)

CUPELLATION is well known to be one of the most effective methods of separating silver and gold from base metals and other impurities, as well as one of the most accurate means for their estimation. In the latter application it consists in absorbing in some porous medium the fused lead oxide formed by the oxidation of the lead "bath," which carries with it the oxides of other base metals present.

It has long been recognized that small amounts of both silver and gold were removed and absorbed together with the base metals, as well as being to some slight extent volatilized, and in work of the highest accuracy a correction is made for the "cupellation loss."

Thus in the assay of bullion this is done by means of "proofs" of like composition, which are manipulated throughout in precisely the same manner as the assay pieces, and are assumed to lose or gain like amounts. In the case of certain very rich ores and commercial products the loss is adjusted by assaying the cupel itself and recovering the precious metal absorbed, but this doubles the labor and does not remedy the loss by volatilization. Arbitrary tables have been drawn up by various authorities with the object of enabling a correction to be applied without the use of a check. These have been of very limited practical use owing to the fact that a number of factors affect the losses sustained.

For a fixed amount of precious metal the loss varies with the amount of lead used, with the nature and amount of the impurities, with the porosity of the cupel, with the air supply, and above all with the temperature at which cupellation is carried on. Pure silver loses relatively much more than gold, but the loss in either is diminished by an addition of the other metal. The concentration of precious metal in the litharge increases as the concentration in the lead increases, but little work has been published connecting the two. If all other conditions remain the same the actual total loss increases, but the percentage loss diminishes, with an increase in the weight of precious metal treated.

The results of an immense number of experiments have been published at various times during the past hundred years giving the actual or the percentage losses suffered under various conditions of cupellation, but the data are widely scattered and few attempts have been made to correlate them, so that no definite law has heretofore been shown to exist

governing the relation of weight and loss, although Fulton has published curves covering certain conditions. If such a law were known it would enable a correction to be applied to a button of any weight, by a calculation applied to the loss observed in a proof of an entirely different weight, but cupelled at the same time under the same conditions. It is believed that such a rule is now available.

This empiric rule may be thus enunciated: *When a Given Amount of Silver (or of Gold) is Cupelled with a Given Amount of Lead, under a Fixed Set of Conditions as to Temperature, etc., the Apparent Loss of Weight Sustained by the Precious Metal is Directly Proportional to the Surface of the Button of Fine Metal Remaining.*

Probably the calculation should be based on the original weight of precious metal taken, but in practice we have to depend on the weight remaining. Variations in the amount of lead have comparatively little influence, and temperature conditions in a given row across a muffle are nearly uniform, so that for practical purposes the proportionality holds good for any one row in a cupellation run, provided the amounts of lead do not differ extremely. Most other variations have smaller effects. If the above is true the following must also be true.

The Loss of Weight Varies as the $\frac{2}{3}$ Power of the Weight, or as the Square of the Diameter of the Button.

The Percentage Loss Varies Inversely as the Diameter of the Button, or Inversely as the Cube Root of the Weight.

Examination of a large number of experimental results, as well as the published data of others, proves that the last proposition is a close approximation to the truth, and therefore proves the others. The readiest proof of the rule, which avoids all calculation, is by the aid of logarithmic paper—that is, paper on which the co-ordinates are plotted on a scale like that of a slide rule.¹ In place of using the ruled paper the distances may be actually laid off by means of a slide-rule scale. When plotted in this way the graph of any equation of the character $y = Cx^n$ appears as a straight line, and all such lines for a given value of the power n are parallel. If the horizontal and vertical scales are equal (as in Figs. 1 and 2) the tangent of the angle made by the line with the x axis is the exponent n .

If therefore we plot on the x axis the weights of a series of silver buttons cupelled under precisely similar conditions, and the corresponding losses on the y axis, and the corresponding points fall approximately on a straight line, then we know that y varies as some power of x . Drawing a line parallel to the general run of these points from the point whose co-ordinates are $y = 1$, $x = 1$ to $x = 10$ or 100 , will indicate at once what the value of n actually is.

¹ Multiple logarithmic paper may be obtained at a very moderate price through the Department of Civil Engineering, University of Wisconsin.

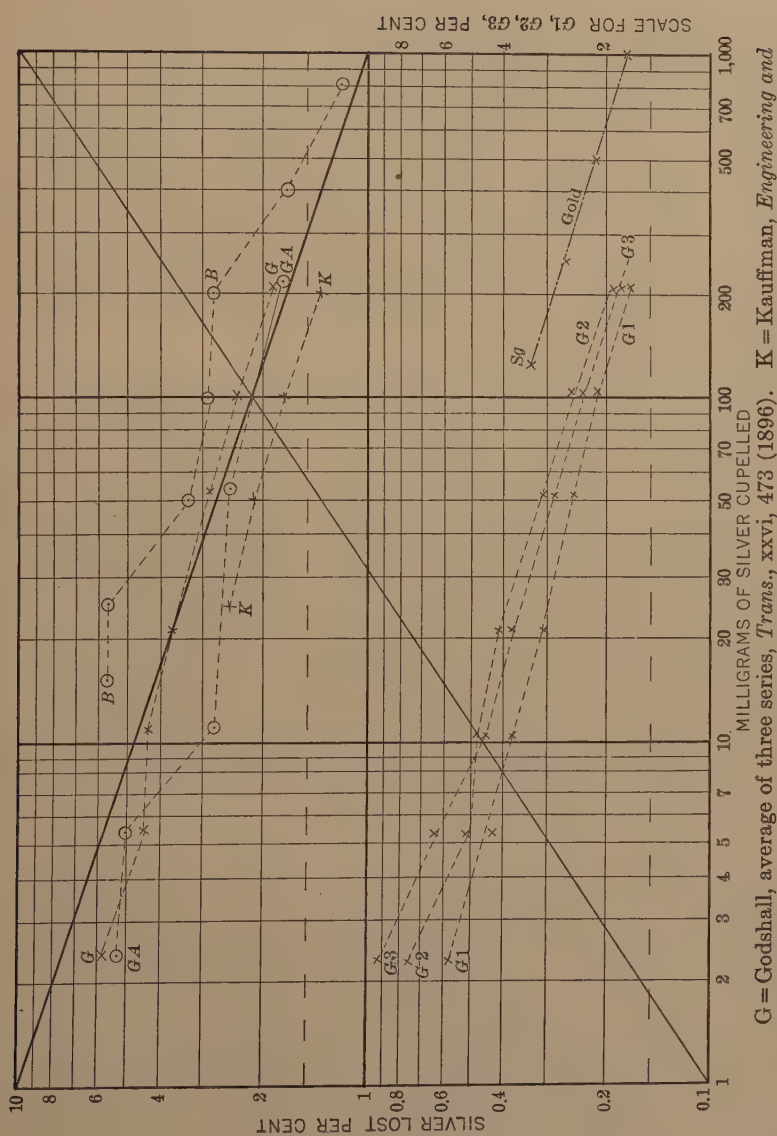


Fig. 1.

The percentage losses alone are given, this being the form used in most publications. Godshall's results for various muffle rows are placed for clearness in the lower portion of the diagram.

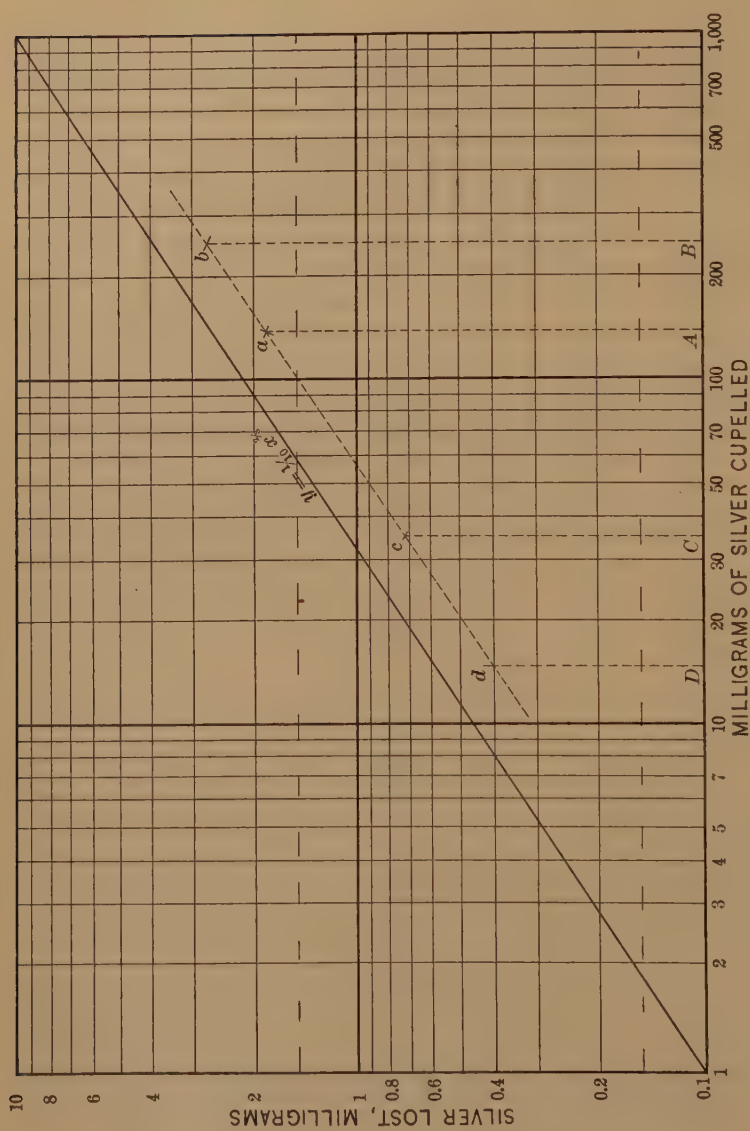


Fig. 2.

If any one will take the trouble to average and plot a series of cupel losses on buttons of various sizes carefully cupelled in the same row, he will, I think, find that they will fall nearly on² lines parallel to

$$y_1 \text{ (weight lost)} = x^{2/3}$$

and

$$y_2 \text{ (percentage lost)} = x^{-1/3}$$

The distances between the parallel lines obtained in various series simply correspond to differences in the values of the constant C for different conditions of temperature, muffle draft, etc.

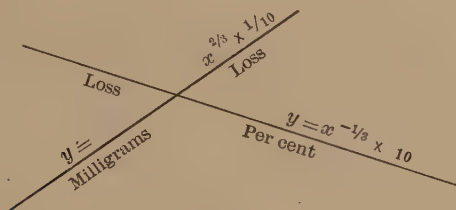


FIG. 3.

A few results are shown thus plotted in Fig. 1.

I have found that such series as those published by Klasek, Godshall, Beringer, and Kauffman (omitting certain erratic ones), and a transcription of the curves plotted by Fulton, all give points falling nearly into general parallelism with the lines indicated, and my own values for silver alone and gold alone agree well except for the smaller weights, where irregularity naturally occurs. This rule is the dominant factor in cupellation loss, temperature alone excepted.

As to the possible application of this rule: An approximate correction may be obtained for an ore or the like, if one proof, as near the expected weight as possible, is cupelled in each row, and its exact loss noted. The corresponding point is plotted on the diagram (a in Fig. 2, A representing the final weight), and a straightedge is placed on the diagram, or a line drawn through a , parallel to the guide line, $y = x^{2/3}$. Noting the points on the line or straightedge corresponding to the weights B, C, D , of other buttons weighed, the correction for each is read at once from the scale. In the example plotted in Fig. 2 a button of 140 mg. weight has lost 1.8 mg.; from this it appears that other buttons of 250, 35, and 15 mg. will respectively have lost about 2.85, 0.72, and 0.40 mg.

In a future paper I hope to present some further data on the cupellation of mixed metals, their relative rates of removal, and their concentration in various portions of the cupel.

² In the case of the smaller weights, say those below 10 mg., a considerable deviation from a straight line will probably be observed, unless a large number are averaged, as the relative error due to a difference of a few hundredths of a milligram then becomes serious.

DISCUSSION

FREDERIC P. DEWEY, Washington, D. C. (communication to the Secretary*).—Even if the data given should be considered sufficient to support properly the empirical rule as stated, this rule would be of extremely limited application in miscellaneous assaying. The fixed conditions must include equality of gold and silver where mixed beads are concerned, owing to the wide variation in the specific gravity of gold and silver. Also, the fixed conditions must include the presence of the same amounts of other base metals besides lead. Such an agreement is often lacking even in the beads from two fusions of the same ore. In practical work it is sufficiently general to be called universal that cupellation beads are not pure precious metal. An apparent loss in cupellation is an algebraic sum and in many cupellations the gains exceed the losses, especially with large beads where the gains lie so completely in the judgment of the cupeller.

For several years I have been engaged in an investigation into the conditions affecting the accuracy of gold-bullion assaying, and have just completed an examination of cupels which included the assaying of 10,000 used cupels. The grand result of the cupel work is an emphatic warning against drawing rigid conclusions regarding cupellation. It is so easy to say "If all other conditions remain the same," but it is extremely difficult, and in practical work impossible, to maintain equal conditions. Cupel absorptions, which constitute such a large proportion of cupellation losses, and in many cases the only loss, may vary considerably in different parts of the same row of cupels, even in carefully regulated work.

* Received Sept. 1, 1915.

Important Topping Plants of California

BY ARTHUR F. L. BELL,* SAN FRANCISCO, CAL.

(San Francisco Meeting, September, 1915)

PRIOR to 1908 the oil production in the State of California had been almost entirely a heavy fuel oil, with a high flash point, but changed within a short period to a large percentage of refining oil with a low flash, by reason of the heavy production of light oil coming in from the Santa Maria field, the Midway field, and the increased production of the Fullerton field.

This is clearly shown in the following tabulation of California's production:

Year	Barrels	Below 20° Bé, Per Cent.	Above 20° Bé, Per Cent.
1908.....	48,307,000	90	10
1910.....	77,698,000	80	20
1912.....	90,075,000	69	31
1914.....	103,600,000	47	53

The bringing in of these high percentages of light oil with their dangerous low flash and the increasing demand for the refined products necessitated the devising of some cheap and efficient means of quickly separating the lighter elements either in the field or at the main distributing points; also it became necessary to remove the excessive percentage of water which many of the light oils carried in emulsion.

UNION OIL CO.

Port Harford Plant

Probably the first important attempt to top light oil in California by other means than the ordinary stills was tried about 1908 by the Union Oil Co., at its plant at Port Harford, San Luis Obispo County, with Santa Maria oils. The process was the invention of Hubert G. Burroughs who afterward obtained a patent for the process, No. 998,837, dated July 25, 1911. The apparatus used consisted of a series of 12-in. pipes mounted in units of three pipes, one above the other, the crude oil being admitted to the upper pipe and filling it about two-thirds of its

*Chief Engineer, Associated Oil Co.

diameter at which level it overflowed to the next pipe below and in the same manner the oil again flowed to the third pipe. Super-heated steam was the heating agent, which passed through endless coils in the lower half of the 12-in pipes. The oil was heated to about 200° F. in the upper pipe and reached a final heat of about 325° F. in the bottom pipe. The vapors generated passed off through the vapor line to the condenser.

When the plant started the gravity of the Santa Maria crude ran from 23 to 24 Bé, and the oil averaged only from 2 to 3 per cent. water, but since then the average percentage of water has increased to 25 to 30 per cent.

This plant was operated for two or three years but the system developed several objectionable features. It was not economical, and by reason of the increase in water contents, the capacity was limited and was therefore abandoned.

Avila Plant

A new plant was designed and erected under the supervision of E. I. Dyer, Engineer in Chief of the Union Oil Co. This plant is at Avila, near Port Harford, San Luis Obispo County, about 40 miles from the Santa Maria field, the oil being delivered to the plant from the field by two pipe lines; one 6 in. and the other 8 in. Although erected several years ago, it is still one of the best designed topping plants on the Coast, and is particularly adapted to topping wet Santa Maria oils. It is topping oils in some cases carrying as high as 20 per cent. water at the same cost for fuel and labor as some other plants topping oils carrying only 2 or 3 per cent. water.

The plant has a capacity of about 10,000 bbl. per day of clean oil or, as the oil is now running, 8,000 bbl. of net oil carrying about 20 per cent. water. The oil coming from the field carries as high as 30 per cent. of water. In connection with the topping plant there is an electric dehydrating system installed by the Petroleum Rectifying Company of California. Its capacity, however, is not enough to handle all the oil from the field, so only as much oil as it will treat is put through it, reducing the water in the oil treated to about 5 per cent.

The treated oil is then mixed with the rest of the oil coming from the field giving an average of about 20 per cent. of water in the oil handled by the topping plant.

The heat used in topping is steam of 150 lb. pressure generated by a battery of five 300-hp. Stirling boilers in a separate boiler house.

There are 12 cylindrical stills mounted in a steel-frame construction in a separate building, the stills being mounted in 2 rows of six each, one row above the other. Those in the upper set are known as the low-pressure stills, those in the lower as the high-pressure stills. Live steam is admitted to the lower row only (Fig. 1.)

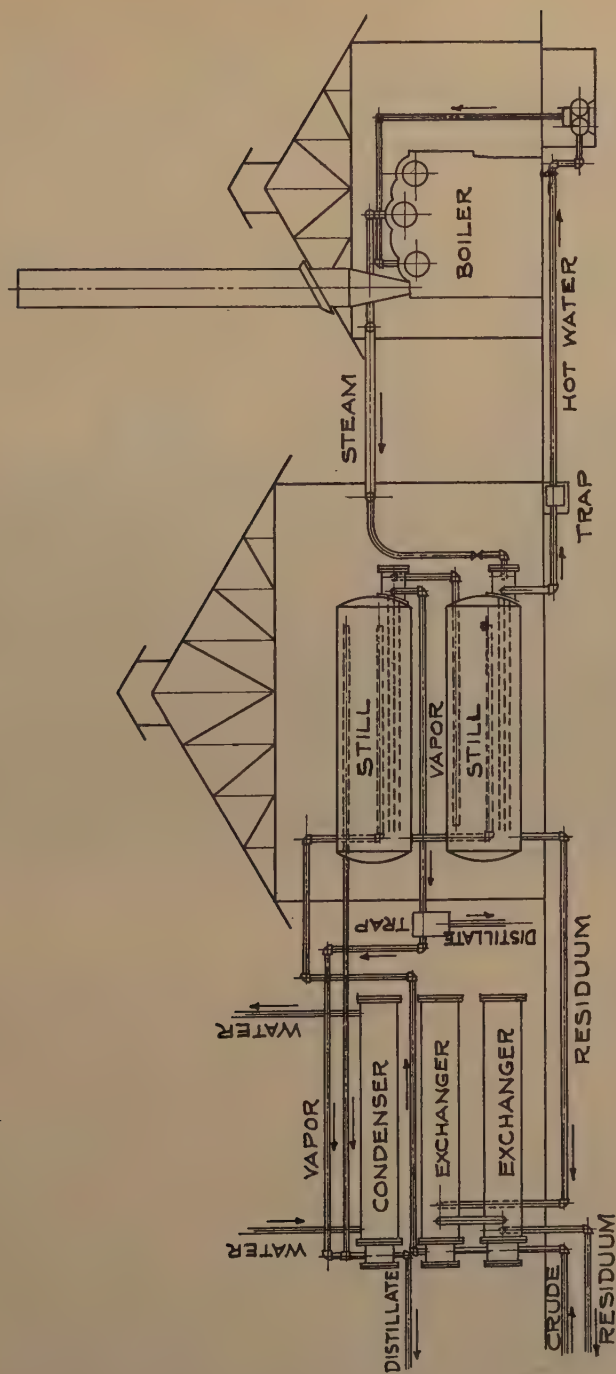


FIG. 1.—UNION OIL CO. PLANT, AVILA, SAN LUIS OBISPO COUNTY, CALIFORNIA.

Each still is about 6 ft. in diameter and 20 ft. long and has mounted on its front end, a steam chest, having three separate vertical compartments; the live steam enters the steam chest of the high pressure still, being admitted to one compartment and passing to the next through a series of 1-in. continuous pipes which travel from the initial side of the steam chest, the full length of the still and return to the next compartment, and in the same manner from the second to the third compartment, so that there is no connection of live steam between any of the compartments of the steam chest except through the heating coils. The condensed water is trapped out of each compartment so that the minimum quantity of condensed water travels through the coils.



FIG. 2.—GENERAL VIEW OF THE AVILA PLANT.

As the steam is condensed, it drains to traps and is returned to the boilers at 280° F. No live steam is admitted to the upper tier, or low-pressure stills, the heating agent being the evaporated vapors from the bottom still which rise and enter the steam chest of the upper still and pass through in the same manner as the live steam entered the lower still. The vapor pressure in both the lower still and inside the coils of the upper still ranges from 40 to 50 lb., the pressure being maintained at a point where the radiation of the latent heat through the coils to the liquid in the upper still is equalized by the heat vapors rising from the lower still. As about half of the vapor is steam, the temperature corresponds closely to that of saturated steam of the same pressure. No vapor leaves the coils of the upper still; nothing is trapped off but the condensed liquids which go to the same condenser as the vapors from the low-pressure still. By this means every degree of latent heat in both the oil and water vapors rising from

the high-pressure still is absorbed by the oil and water evaporated from the low-pressure still, thereby effecting a saving of about 50 per cent. of the fuel necessary to top the oil in ordinary stills. The vapors from each low-pressure still flow into the condenser belonging to that battery.

There are six separate condensers mounted on the top of a steel construction outside of the steam-still house, each connecting to a battery of a high- and a low-pressure still. The general design of the condensers is rectangular with a vapor chest at the rear end of the same general design as that of the stills, except that the dividing partition in the chest is horizontal. The vapors are admitted to the upper portion of the chest and pass through a series of 2-in. return pipes, surrounded by the con-

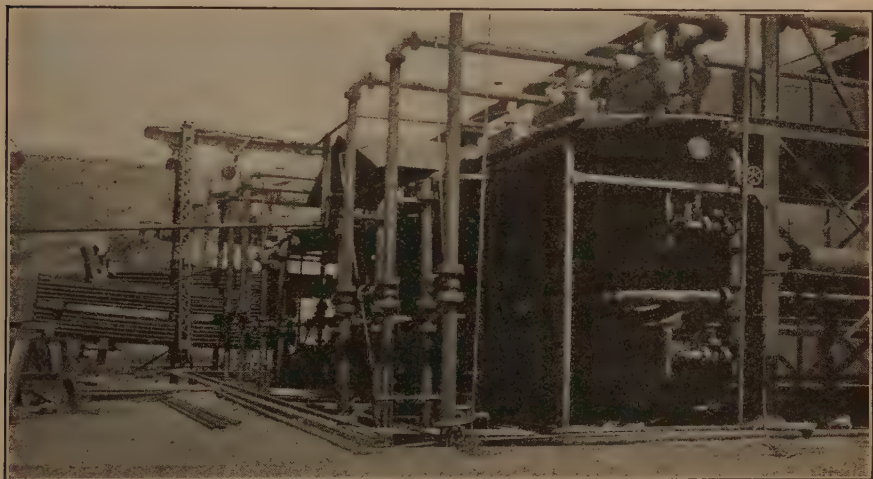


FIG. 3.—COOLERS AND RESIDUUM HEAT EXCHANGERS. AVILA PLANT.

densing water, and return to the lower portion of the chest, from there draining to the proper storage tanks.

Mounted below each condenser are two heat exchangers set one above the other. These heat exchangers are similar in construction to the condenser except that they have 1-in. return pipes, and have two vertical dividing partitions in each chest; also heated residuum surrounds the pipes instead of water. The heated residuum from the high-pressure still passes into one end of the body of the upper exchanger and travels backward and forward four times, guided by interior partitions. It then passes into the end of the body of the lower exchanger, again traveling backward and forward four times before passing out as finished residuum to storage. In all condensers and exchangers, the liquids counterflow.

The crude oil in taking up the heat in the exchangers comes into one side of the chest of the lower exchanger, passes backward and forward twice through a series of 1-in. return pipes and then is discharged from

the opposite side of the chest to the upper exchanger. It passes through two series of return pipes in the upper exchanger, leaving the chest on the opposite side from where it entered. It flows to the low-pressure still, and after giving off a proportion of its vapors of oil and water, is pumped to the high-pressure still where all of the remaining water and distillates are taken off. The residuum oil then passes to the exchangers.

When I examined these stills, the pressure of the vapors rising from the low-pressure stills was 3.6 lb. with a temperature of 183°F. In the high-pressure stills, and inside the coils of the low-pressure stills, the vapor pressure was 48.3 lb. with a temperature of 279° F.

This is a most economical system, since the only steam admitted to the plant is through the high-pressure stills; all of the work done in the low-pressure stills is therefore an absolute saving.

The plant is running on what is known as double effect, but at times has been run on triple effect, that is, live steam was introduced into the first still only and the heated vapor from the first still under pressure was admitted to the coils of the second still, and the heated vapor from the second still, necessarily under a less pressure, was admitted to the coils of the third still, the crude oil flowing first to the low-pressure, then to the intermediate, and finally to the high-pressure still.

Distillation by Double Effect

	Water, Per Cent.	Distillate, Per Cent.
Evaporation from low-pressure still	56.95	34.45
Evaporation from high-pressure still	42.05	64.15
Evaporation from interchanger and unaccounted.	1.00	1.40
	<hr/> 100.00	<hr/> 100.00

Triple effect, as would be expected, showed a greater saving than the double effect, but reduced the quantity of crude that could be put through in a given time.

Distillation by Triple Effect

	Water, Per Cent.	Distillate, Per Cent.
Evaporation from low-pressure still	55.80	37.42
Evaporation from intermediate still	29.75	31.36
Evaporation from high-pressure still	13.42	29.74
Evaporation from interchanger and unaccounted	1.03	1.48
	<hr/> 100.00	<hr/> 100.00

From the above figures, it will be seen that with the double-effect arrangement about 50 per cent. of the fuel is saved, and for triple effect about 65 per cent., over a single system of evaporation.

When observed by me, the average temperature of the crude entering the heat exchangers was 61.67° F.; leaving the heat exchangers before entering the low-pressure stills, the temperature was 141.45° F., showing a saving by rise in temperature due to exchangers of 79.78° F.

The economy of the plant is shown by the following:

Duty Records per Barrel of Dry Fuel Oil Used for the Last Three Months of
1914

Barrels of crude oil treated.....	63.35
Barrels of water actually removed from crude.....	10.67
Barrels of residuum produced.....	42.34
Barrels of distillate produced.....	8.54
Barrels of residuum pumped from plant.....	43.80
Barrels of distillate pumped from plant.....	9.52
Barrels of circulating water pumped.....	156.46
Barrels of miscellaneous oil pumped.....	9.08
Kilowatt hours—lighting, power, etc.....	2.53

Thermal Duty of 1 bbl. of Dry Fuel Oil

	B.t.u.
Evaporating water from crude.....	4,135,984
Evaporating distillate from crude.....	425,128
Kilowatt hours—lighting and power.....	35,940
Total useful duty.....	4,597,052

The average percentage of water in the crude was 18.71 per cent.

A recent test run on double effect showed the following results:

Fuel consumed per net barrel of:	Per Cent.
Crude treated.....	1.75
Residuum manufactured.....	2.51
Distillate manufactured.....	11.70
Water evaporated.....	11.58
Both water and distillate evaporated.....	5.82
Water in the crude oil.....	15.38

This system is ideal for Santa Maria oils and entirely eliminates the coking which takes place when direct heating is used under ordinary stills or retorts. The water carried by the oil is as salty as sea water and when evaporated throws down salt crystals in the stills. In the case of fire-heated stills, this salt mingles with the coke and forms a hard incrustation, which necessitates closing down frequently and chipping out the coke; whereas with this form of still, when salt crystals are formed, the operators employ the unique method of shutting down the stills about once a week, emptying the stills of oil and filling with hot water, thus dissolving the salt and passing it off by pumping a continuous stream of heated water through

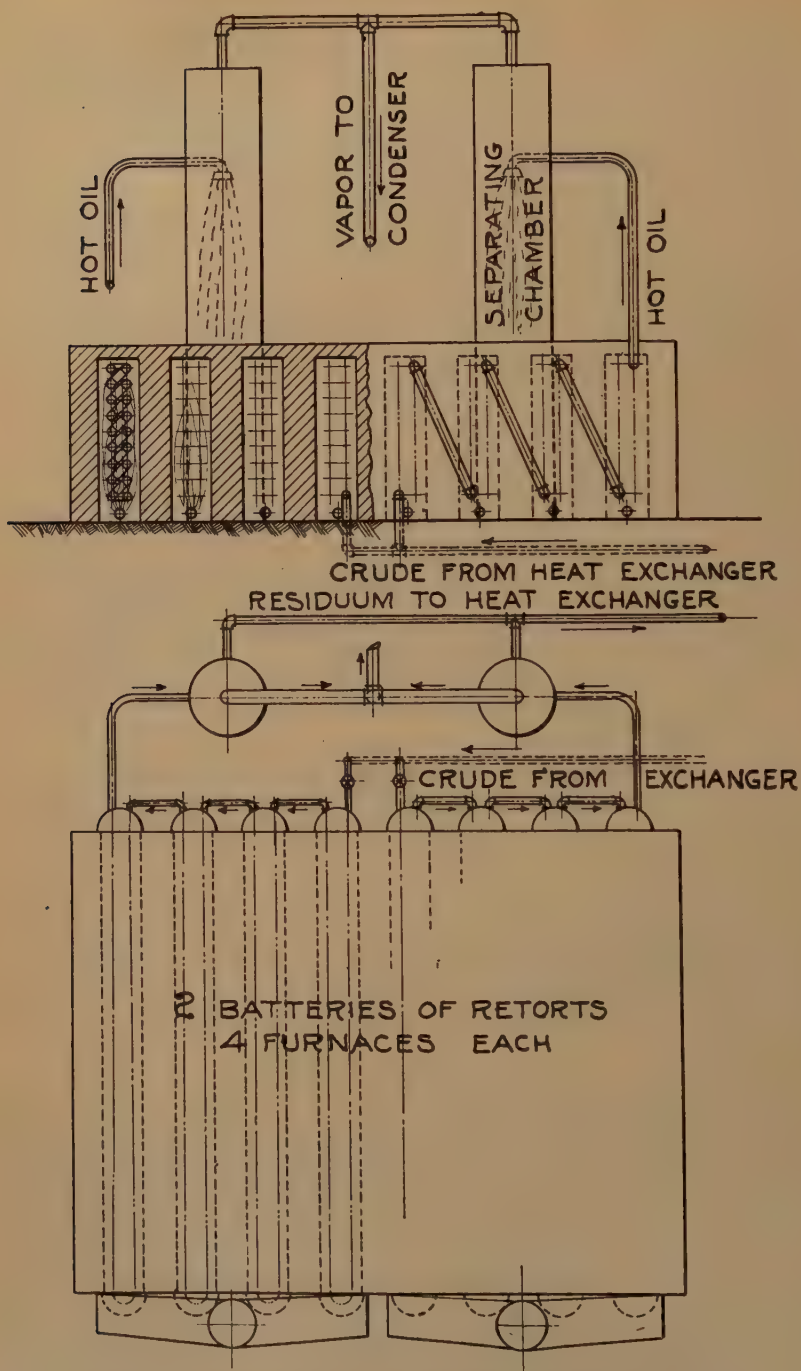


FIG. 4.—TOPPING PLANT AT BREA REFINERY, UNION OIL CO.

the stills. Two patents on this system have been applied for by Mr. Dyer and are now pending in the patent office.



FIG. 5.—FRONT VIEW OF BREA TOPPING PLANT.



FIG. 6.—REAR VIEW, SEPARATORS IN BACKGROUND.

Brea Refinery

The Union Oil Co. has two other topping plants in the Fullerton field, near Los Angeles. The most important is the Brea refinery. The first

portion of this plant was completed in August, 1911, to handle the light oils from the Birch well, which came in with a very gassy oil flowing about 2,400 bbl. per day. It consists of eight 40-hp. drilling boilers, with half the tubes removed; these boilers were set in two rows of four facing each other with an alley between, the oil being run through each battery of four boilers continuously. The plant when first started treated successfully about 100,000 bbl. per month of clean dry oil carrying not more than 0.5 per cent. of water, removing 22 per cent. of 53° tops. However, it has since developed that this plant is not adapted to handling wet oils and will



FIG. 7.—REARRANGED FURNACE FOR OIL BURNING. BREA REFINERY.

not work economically with oils carrying more than 2 per cent. water. It is therefore kept running on the cleaner oils.

An addition to the topping plant was built in March, 1913. This consisted of two batteries of retorts, each battery being divided into two separate divisions, each division having four furnaces and each furnace carrying a double row, 9 pipes high, of 3-in. return pipes, 20 ft. long. Each division of the battery is independently fed with crude oil. The oil admitted to the bottom pipe of the furnace travels up through the nine coils, coming out the top and returning to the bottom pipe of the next furnace, again leaving the upper pipe and so on through the four furnaces. The heated oil from each side of the battery flows by independent pipes

to one of the separators. This makes each 8-furnace battery and one separator a single unit.

The vapors are then led from the separating chambers to the condensers and on to the storage tanks.

The residuum is taken from the bottom of the separators, passed through the heat exchangers and on to the residuum storage.

The coils in the furnace are heated by gas fed through perforated pipes running the full length of the furnace. It was found that this form of furnace is not well adapted to oil firing. A change was, therefore, made in one of the batteries, the coils being mounted horizontally in a brick furnace, allowing plenty of combustion room below. This furnace is working very satisfactorily and it is the intention to change the other furnaces from vertical to horizontal position so as to be adapted to oil firing. (Fig. 7.)

The total capacity of the two batteries of retorts with gas fires was 60,000 bbl. per month of 21 to 22° Bé. oil, carrying 6 per cent. B. S. & M., from which was topped about 10 per cent. of 51 gravity distillate, giving a residuum of 18° Bé. and from 160 to 180 flash. With the change in construction and the use of oil firing, it is believed the capacity will be doubled.

The hot oil at the outlet end of the retort is choked down to a pressure of 40 lb., but flows into the separator at practically atmospheric pressure. This choking of the oil gives a back pressure at the inlet end, due to friction in the retorts, of 80 lb.

The crude oil leaves the heat exchanger and enters the first coil of the retort at 180° F.; it leaves the coils and enters the separators at about 450° F.

This refinery not only tops oil but turns out finished gasoline, but as the purpose of this paper is to describe topping plants only, no description will be given of the other portions of the plant.

Naranjal Topping Plant

The Naranjal reduction or topping plant of the Union Oil Co., is also located in the Fullerton field. This is the smallest of the three Union topping plants, having a maximum capacity of 2,200 bbl. per day. The oils treated are 16° to 18° Bé. and tops recovered 56° Bé. The plant has been built about 4 years and is known as a Brown-Pickering topping plant, it having been designed by A. H. Brown, the Superintendent. (Figs. 8, 9, 10.)

The plant consists of two independent units, each unit having a rectangular still about 4 by 4 by 20 ft. with cover bolted on and bottom of still fitted with cross baffles. The still is set, with a fall of about 6 in., over a brick furnace in which are placed 13 3-in. retort pipes, having

return bends so as to make a continuous coil. The hot gases from the still furnace pass through an underground flue to a 40-hp. drilling boiler,



FIG. 8.—SIDE VIEW OF THE NARANJAL PLANT, SHOWING BOILERS AND STILLs.

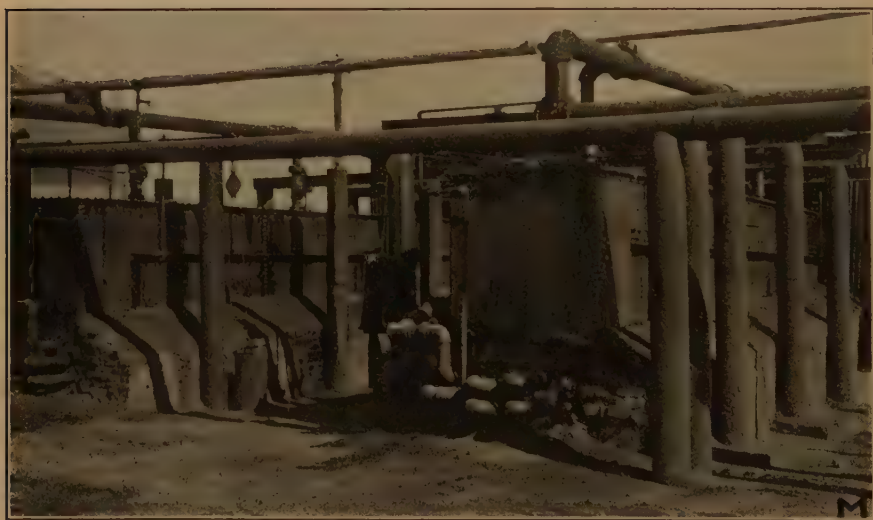


FIG. 9.—STILLS. NARANJAL TOPPING PLANT.

bricked up in the ordinary manner, these gases passing first under the boiler then through the tubes and out of the stack. In addition, the ordinary field type of heat exchanger is used, made of a small pipe

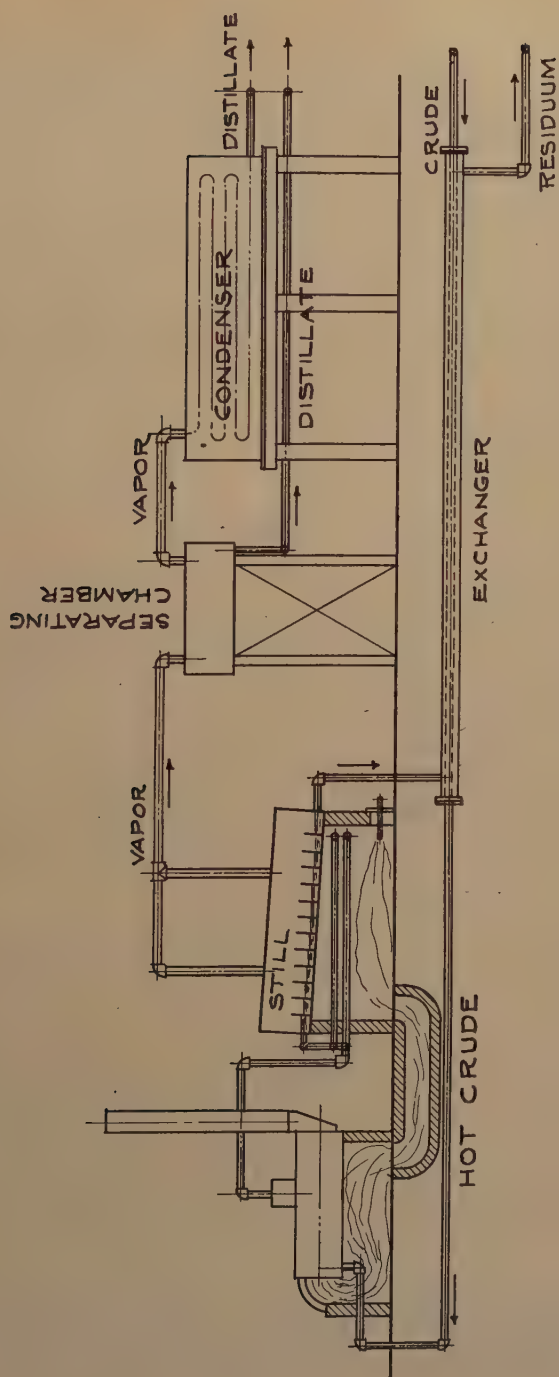


FIG. 10.—PLANT OF THE UNION OIL Co., NARANJAL, FULLERTON FIELD. BROWN-PICKERING DESIGN.

inside of a large one; for cooling the vapors, a series of wooden condenser boxes with pipe coils inside are provided.

In operating the plant, the crude oil is pumped through the inner pipe of the heat exchanger, the hot residuum flowing between the two pipes. The partially heated crude oil passes through the boiler, heated by waste gases, and from there through the coil retort in the furnace under the rectangular still, then into the high end of the still, where it is released and allowed to flow back and forth guided by the baffles until it reaches the lower end of the still, from which it flows by pipe through the heat exchanger to the residuum storage. As the oil flows through the still, which is heated by fire on its under side, the vapors pass off to a separating chamber, where the heavier vapors condense and are drawn off, and the lighter vapors pass to the condensers.

Each unit is a duplicate of the other, but is run entirely separate up to where the vapors enter the separating chamber; from there on the mixed vapors pass to the condensers. This system carries the notice "Patent applied for."

A number of these Brown-Pickering plants have been constructed in the oil fields and have given fair results with clean oil. They are not well adapted for wet oil, for as soon as they are worked to capacity with wet oil, foam appears, which discolours the distillate, and the construction of the still does not admit of putting any pressure on it to hold down the foam.

STANDARD OIL CO.

On the Murphy, Coyote Hills, property, Fullerton field, since bought by the Standard Oil Co., is one of the Brown-Pickering topping plants, consisting of four rectangular stills about 4 by 4 by 10 ft., each set with a gradual fall so that the oil admitted into the upper end of the first still flows down to its lower end and out into the upper end of the next still, which is set beside it and so on until the oil had flowed by gravity through the four stills.

At the rear end of the four stills is erected one 40-hp. boiler so constructed that the hot gases from the four still furnaces pass under the boiler and through the tubes and out of the boiler stack. The vapors from the first two stills were run together and kept separate from the vapors coming off the third and fourth stills which were also run together. The vapors were then each run through separate pipe heat exchangers, the vapors in the inside pipe and the crude between the inner and outer pipes.

The continuation of the inside vapor pipe was then returned toward the plant and surrounded by an outer pipe this time filled with water, this forming the condenser for the vapors. The condensed vapors were then run to storage tanks.

In operating, the oil was first pumped through the outside of the vapor heat exchanger, then through a residuum heat exchanger, then to the boiler and from there to the upper end of the first still, then out of the lower end of the last still to residuum storage.

This plant had a capacity of about 1,200 bbl. per day of clean oil and 600 to 650 bbl. of wet oil carrying 25 per cent. water. If this latter capacity was exceeded, foaming took place.

With the crude oil entering the vapor heat exchanger at about 60° F., it entered the boiler heater at 80°, and left the boiler to enter the first still at 90° F.

The temperature of the oil on leaving the four stills ran 175° F., 200° F., 220° F., and 240° F., respectively. The fuel consumed was gas.

After the Standard Oil Co. purchased the property, the plant was used for a time as a dehydrator. The company has since erected a 6-treater electric dehydrator, installed by the Petroleum Rectifying Co., which cleans the oil more satisfactorily and at a much less cost.

RICE RANCH OIL CO.

Another Brown-Pickering plant of the same type, but of a single unit, has been erected by the Rice Ranch Oil Co. in the Santa Maria field, Santa Barbara County. This plant is used mostly as a dehydrating plant but has also been used for topping. It was erected about 1911 and has worked very satisfactorily. Its capacity is claimed to be about 650 bbl. of crude carrying 3 per cent. water, but it also handles, at a reduced capacity, oil carrying as high as 30 per cent. water. The course of the oil through the plant is the same as in the other Brown-Pickering plants.

AMERICAN OIL FIELDS CO. OF CALIFORNIA

Another important topping plant is that erected by the American Oil Fields Co., of California, about 1910, on section 36-31-22, in the Midway field. This plant has an economical capacity of about 18,000 bbl. of crude per day. It is arranged with three batteries, each battery consisting of eight independent retorts, making 24 retorts used on crude oil. Each battery delivers its heated oil to a single separator, where the vapors are separated, the vapors being led to a common header which distributes them to a series of independent condensers. (Figs. 11, 12.)

This plant is well constructed and economically run and the company is to be complimented for its neat appearance.

All the condensing water is saved and run back through four independent cooling towers, which have a remarkable efficiency, since the circulating water after passing through these towers is cooled far below the temperature of the atmosphere. When I took observations, the water



FIG. 11.—PLANT OF AMERICAN OIL FIELDS CO., SHOWING FRONT OF RETORTS.



FIG. 12.—REAR VIEW OF AMERICAN OIL FIELDS PLANT, WATER COOLING TOWERS IN BACKGROUND.

entered the towers at 140° F. and was taken out of the bottom at 65° F. with the surrounding temperature 99° F.

The plant is not only well adapted to topping clean crude, but is often called on to handle oils carrying as high as 25 and 30 per cent. water.

The fuel used at the refinery is natural gas.

SANTA FE RAILROAD CO., FELLOWS

The first really successful Trumbull plant, and the one that first attracted attention was erected at Fellows, Kern County, for the Santa Fe R. R. Co., to top its light oils so as to make safe fuel for locomotives. This plant consisted of two units each of six 12½ in. by 20 ft. retorts, both units discharging into the one vapor separator.

The records of this plant for January, 1912, showed for 31 days' operation, with one of the units off for 5 days:

	Barrels	
Crude delivered to the plant.....	120,760.67	21° B ₆ .
Tops taken off.....	14,772	12.25 per cent.
Gasoline recovered.....	36.60	0.03 per cent.
Fuel consumed.....	1,575	1.34 per cent.

The Santa Fe R. R. Co. has since contracted its light oils to the General Petroleum Co. and has, therefore, closed down this plant.

PACIFIC CRUDE OIL CO., MIDWAY FIELD

The Pacific Crude Oil Co. built a small Trumbull plant on its property near Fellows for the purpose of topping oil in the field. Subsequently the company contracted its oil, so the plant was never started up. A good example of a small Trumbull topping plant is illustrated in Figs. 13 and 14.

GENERAL PETROLEUM CO., VERNON

The General Petroleum Co., in 1912, completed an 8-in. line from the Midway oil fields, Kern County, to San Pedro, Los Angeles County, and as the oils transported were too light and valuable for fuel purposes, it was necessary that a topping plant be erected at or near the terminal. The company adopted the Trumbull system of topping and erected a plant of 15,000 bbl. daily capacity (Figs. 15 to 19A) at Vernon, in the outskirts of Los Angeles. All incoming oil is topped here and the residuum pumped to tidewater at San Pedro. However, while this plant has a rated capacity of 15,000 bbl. daily, it has since been determined that the most economical capacity of the plant is about 12,000 bbl.

The General Petroleum Co., in adopting the Trumbull system, pur-

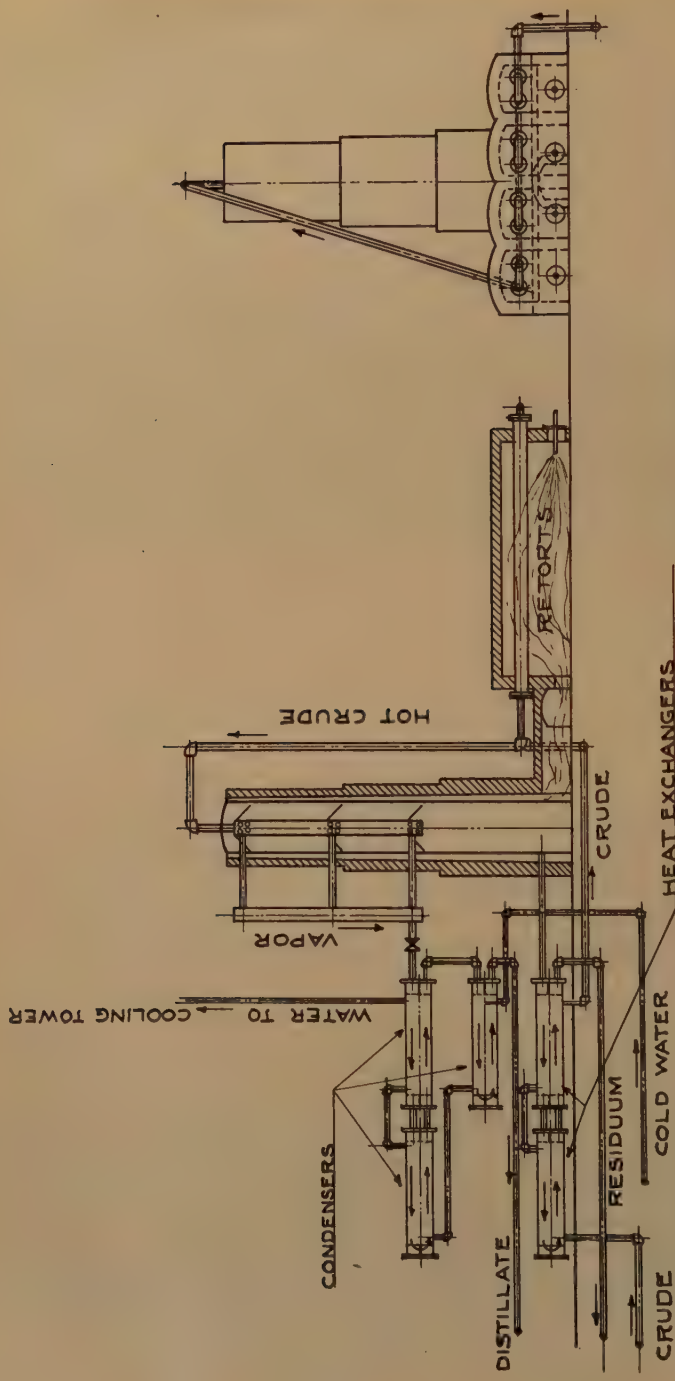


FIG. 13.—SMALL TRUMBULL TOPPING PLANT, PACIFIC CRUDE OIL Co., MIDWAY FIELD, KERN COUNTY, CALIFORNIA.

chased a half interest in the Trumbull patents which to date are as follows: No. 996,736, patented July 5, 1911; No. 1,002,474, patented

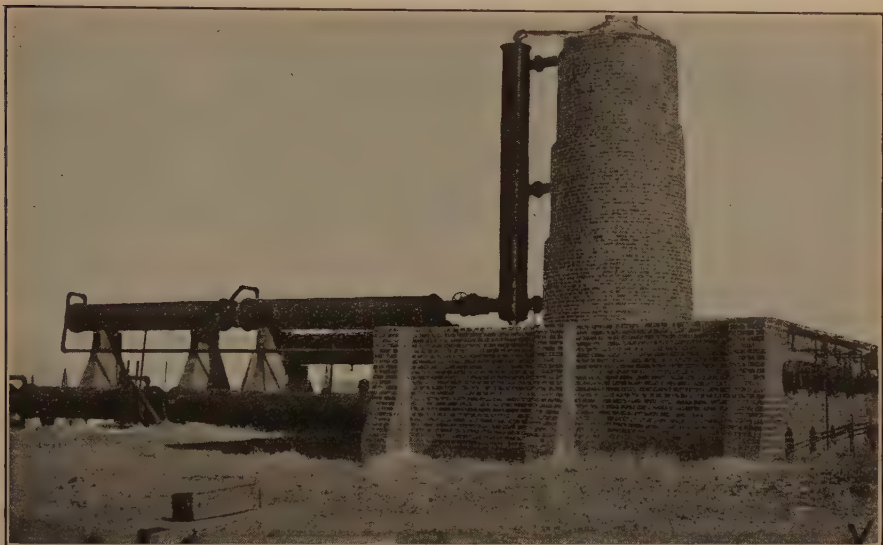


FIG. 14.—TRUMBULL TOPPING PLANT OF PACIFIC CRUDE OIL CO.

Sept. 5, 1911; No. 1,070,301, patented Aug. 12, 1913; also patents secured in seventy-two foreign countries.

The special features claimed for the Trumbull process are:



FIG. 15.—FRONT VIEW OF TRUMBULL PLANT OF GENERAL PETROLEUM CO.

The saving of heat in the evaporating chamber by means of conical spreaders in the evaporators, all oil being forced to run down the inside

surface of the shell, which is heated on the outside by the escaping flue gases.

The condensing of the first-run vapors and the re-evaporation of these

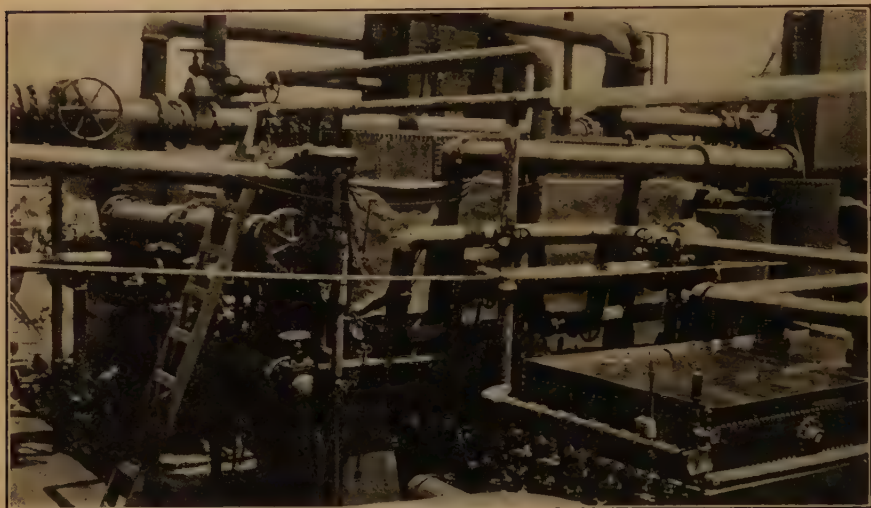


FIG. 16.—SEPARATORS.

distillates by passing them through a specially designed separator, which derives its heat from the outflowing first-run vapors.



FIG. 17.—RETORT.

And the recovery of the re-evaporated distillates from separate compartments of the separator thereby furnishing different gravities of finished products.

The Vernon plant was completed in 1912, at a cost of about \$80,000 and has been in continuous successful operation since that time. The plant consists of two independent batteries, each battery having two retorts discharging into a single evaporating chamber.* Each battery has two separator boxes, where the condensed distillates are re-run into finished products; also three heat exchangers about 4 ft. in diameter with 18-ft. tubes; two first-run vapor condensers of approximately the same size; and four re-run vapor condensers 30 in. in diameter with 18-ft. tubes.

In addition to the condensers, there are water-jacketed coolers consisting of one or two small pipes laid within a larger pipe for the purpose of further cooling the distillates before going to the storage tank.

In this plant, tubular condensers and water-jacketed pipe coolers have



FIG. 18.—VAPOR CONDENSERS AND PIPE JACKETED COOLERS.

been used in every case instead of the ordinary condensers generally used about a refinery.

Since the plant was built, it has been found advisable to make several changes and additions; one important change being in the retorts. The original retort consisted of six 12½-in. pipes about 20 ft. long connected together at their ends with 4-in. pipe returns. Vent pipes were inserted at one end of each retort to lead off gases which, if trapped in the large pipes, would cause unequal heating and sagging of the retorts. It was

* This chamber is called a "separator" on all the plants described except in connection with the Trumbull plants where it is called an evaporating chamber.

found that there was not heating surface enough, the escaping gases in the stack running as high as 1,200° F. Additional 4-in. retort pipes were, therefore, placed above the 12½-in. pipes, thereby increasing the heating surface about 50 per cent., the result being that the temperature of the escaping gases was reduced to about 650° F. (Fig. 17.)

The company has lately torn down one of the retorts and rebuilt it entirely of 4-in. pipe. This change has resulted in a marked improvement.

Another important feature added to the original construction is a large rectangular dephlegmator and re-evaporator erected at each unit through which all the first-run vapors pass. This dephlegmator, it is claimed, has improved the quality of the products by removing the heavier distillates which originally discolored them.

The oils handled average 20° to 21° Bé.; the residuum manufactured, 16° to 17° Bé.; the tops manufactured, No. 1 of both 55° and 58° Bé., No. 2 of 45° Bé. and still bottoms of 35° to 37° Bé.

Special cuts of various gravities are also turned out to order.

*Temperatures Maintained at Plant and Percentages of
Oil and Water*

Crude run to heat exchangers.....	80° F.
Heat exchangers to retorts.....	264° F.
Out of retorts.....	445° F.
Residuum to storage.....	190° F.
Water in oil, per cent.....	0.5
Fuel consumed, per cent.....	1.6

The last item does not include oil consumed in pumping about the refinery, this being done from the main pipe-line pumping station, and by auxiliary electric pumps. The above, however, includes pumping of circulating water through the cooling towers.

The course of the oil through a single battery of the Trumbull plant at Vernon is as follows:

The crude, previously heated by passing through heat exchanger *H* (Fig. 19), is forced through the retort pipes *K*. From there, it passes upward by a single pipe to the top of the evaporator *A*, and is sprayed over a conical spreader which directs the fluid against the inner surface of the shell. In its downward course, this heated crude comes in contact with additional conical spreaders, continuously forcing it against the heated shell. The vapors separate from the hot crude in the evaporating chamber and enter a vertical pipe which is perforated directly under the apex of each conical spreader. This pipe delivers the vapors into three outlets which terminate on the outside in a single down pipe and lead the vapors through the upper separator *B* to the three inner chambers *L* of the recently constructed dephlegmator *D*. The inner

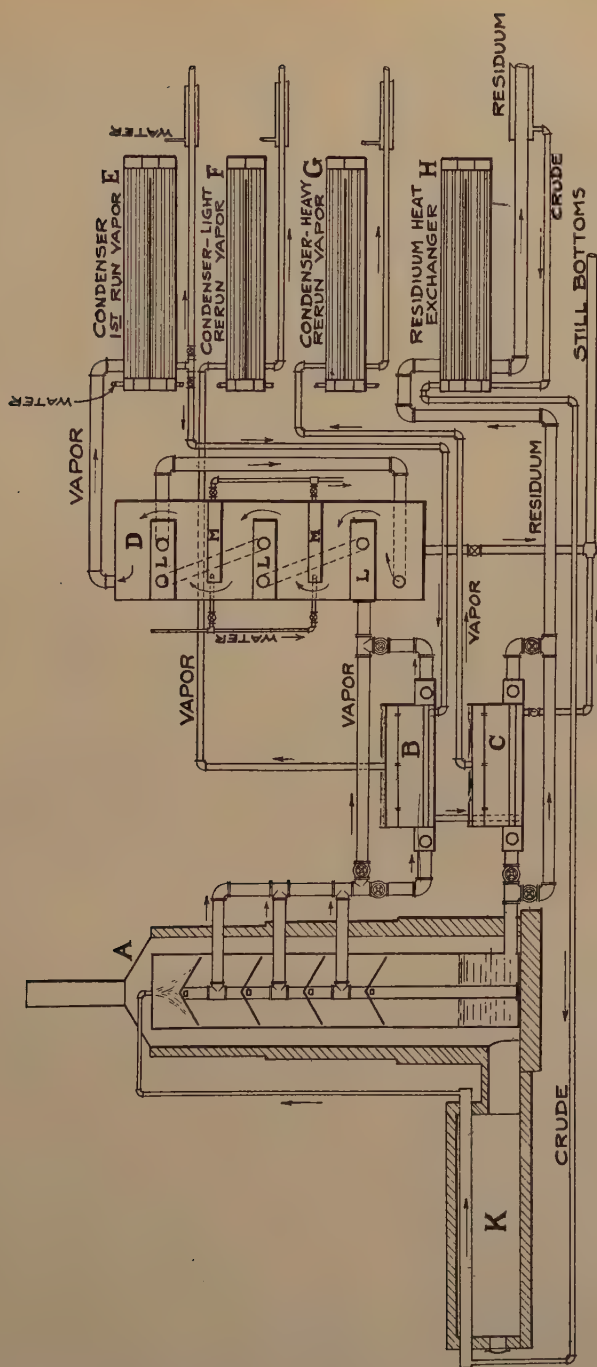
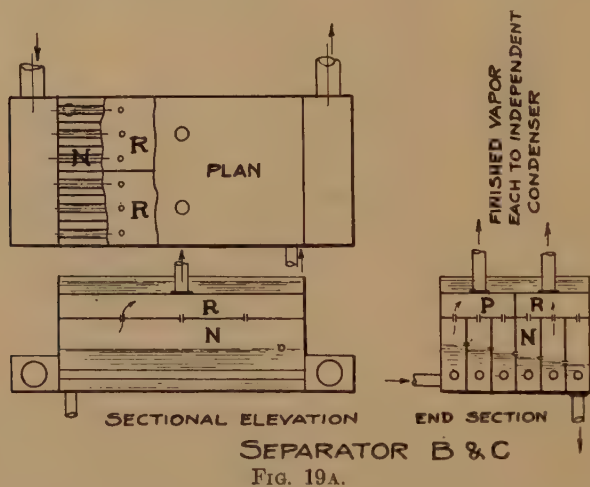


Fig. 19.—GENERAL PETROLEUM Co., TRUMBULL TOPPING PLANT, VERNON, LOS ANGELES COUNTY, CALIFORNIA.

chambers *L* are sealed from the rest of the interior of the rectangular box forming the dephlegmator.

Between each of the three chambers *L*, are two water compartments *M* placed across the box. The incoming vapors enter the lower chamber *L*, pass from the lower to the middle chamber *L*, and from the middle into the upper chamber *L*, and from there through a pipe into the open portion at the bottom of dephlegmator *D*. These gases coming in contact with chambers *L*, which are filled with the hotter vapors, are re-heated.

As the re-heated vapors flow up in the main box *D*, they come in contact with the cooling surfaces of the water compartment *M*, which condense and throw down the heavier distillates on to the heated surface of the chamber *L*, the latter evaporating any of the lighter oil, which might be entrained with the condensed heavier distillates. The result of



this operation, it is claimed, is that all the heavy discolored distillates finally reach the bottom of the dephlegmator, while the light clean vapors pass onward and out of the dephlegmator to a battery of two large tubular water condensers *E*, which are of similar construction to the residuum heat exchangers.

The distillate from condenser *E* flows to the separator *B* entering the bottom compartment at one corner, traveling forward and backward six times through a series of compartments *N*, heated by the hot vapors coming from the evaporator *A*. (Fig. 19 A). As the distillates flow through these heated compartments *N*, they are evaporated, the vapors rising through openings into either of the two upper compartments *P* and *R*, from each of which the vapors are led to two independent condensers *F*.

The outside upper portions of the separators *B* and *C* are water jacketed, and as the hot vapors rise in the compartments *P* and *R*, any heavy

distillate carried by the vapors is condensed and drops back into compartment *N*, so that only the lighter vapors escape from the respective compartments.

The unevaporated distillates pass on to separator *C*, where they are treated in the same manner as in separator *B*, the vapors being conducted to two independent condensers *G*.

On account of the distillates being heavier in separator *C* than in separator *B*, a greater heat is required to evaporate them, which is derived from the hotter residuum instead of from vapors. This arrangement of two separators with four independent compartments gives four separate re-run distillates of different gravities, practically without additional cost for fuel, the heat expended in doing this work being derived from the original heat absorbed in the retorts, except for the steam which is admitted into the separators to help drive over the vapors.

The unevaporated distillate from separator *C* passes on with the heavy distillate from dephlegmator *D* to the storage tanks.

Bypasses are arranged in both the vapor and residuum lines, so that all or only a portion of the vapor or residuum may travel through the separators; by this means, only the required heat is obtained.

The residuum after falling to the bottom of evaporator *A*, passes through the separator *C*, and on to a series of three heat exchangers, each exchanger having a diaphragm in the shell, which guides the oil backward and forward; it then enters the next shell and so on into the third shell from which it passes through water-jacketed pipe coolers to storage.

The cold crude, entering these exchangers through the tube compartment at the end of the shell and by means of three diaphragms, is guided backward and forward through the tubes four times, this process being repeated through each of the exchangers, the crude entering the coldest exchanger and passing out of the hottest.

THE SHELL CO., OF CALIFORNIA

The Shell Co., through an affiliated company known as the Simplex Refining Co., purchased in May, 1915, all the Trumbull topping system patents, at a reported figure of \$1,000,000, and is constructing at Martinez a Trumbull topping plant in connection with a complete refinery.

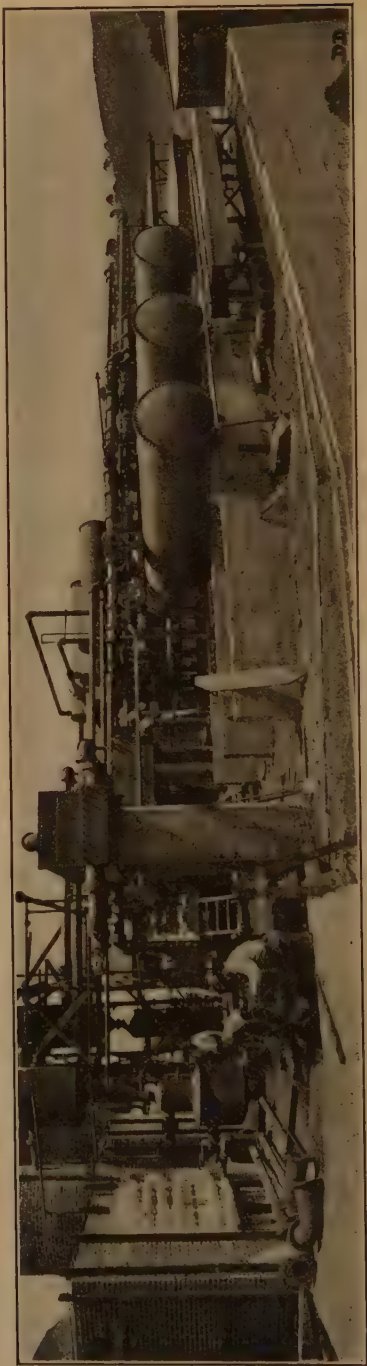
This plant will be supplied by an 8-in. pipe line running from the Coal-inga field to Martinez, where the company has rail and deep-water facilities.

The topping plant now being constructed is a single unit and has an economical capacity of about 5,000 bbl. of crude oil in 24 hr. No money is being spared to make it as complete and economical as the process will permit. In general design, the plant is similar to the General Petroleum



GENERAL PLAN.

SEPARATORS.



SIDE VIEW.

FIG. 20.—TRUMBULL TOPPING PLANT OF THE SHELL CO.

Co.'s Vernon plant. Many improvements have been made; one of the most noticeable is the spreading of the different parts so that they can easily be inspected and repaired; also the change in design of the retorts, and various minor changes which will be to the advantage of the new plant.

Another feature of the Shell plant is the addition of an evaporator column in place of the dephlegmator lately erected at the Vernon plant.

In addition to the main topping plant, the Shell Co. has also erected a smaller Trumbull retort with its evaporating tower for the purpose of running asphaltum and doing other experimental work. This part of the plant is not complete and is not otherwise referred to in this paper.

As at the Vernon plant, all condensing and cooling will be done in tubular condensers.

The process will be carried on as follows:

The cold crude enters first the three vapor condensers *E* and *U* (Fig. 21), then passes forward and backward four times through the tubes of each exchanger, then to the coldest of the residuum heat exchangers *H*, through the five exchangers *H* in the same manner as the other exchangers, leaving the hottest exchanger to enter the top of the evaporator column *T*, which is practically of the same construction as the main evaporator except that the heat is derived from a surrounding vapor jacket. The crude on entering the column *T* is diverted to the heated side in its downward course by the conical deflectors and finally drops into the base of the column. From there, it passes through the retort *K*, then up to the top of the evaporator *A*, where in its downward course, it is again deflected against the shell heated by the furnace gases, and falls to the bottom of the tower.

The first vapors to come off are from the hot crude in the evaporator column *T*, which are led to the separator *B*, passing through the heating tubes and out of the opposite end, then to the condenser *E*, where the distillates are condensed and returned to the lower compartment (*N*) of separator² *B*, the process of evaporation being the same as described in the General Petroleum Co.'s Trumbull plant.

The vapors on leaving separator *B*, which will have the lightest gravity, are condensed in condenser *F* and then pass off through jacketed pipe coolers to storage, while the unevaporated distillates from separator *B* pass on to separator *C*.

The second vapors coming off from evaporator *A* are collected in the central perforated pipe and led to the jacket of the evaporator column *T*. After imparting a certain amount of heat to the inner shell, they pass through the heating tubes in separator *C*, then to condenser *U*, the condensed distillates being returned to compartment *N* of separator *C* to be evaporated as previously described. The vapors then flow to condenser

² For detail drawing of separator see Fig. 19A.

G. These distillates are of a heavier gravity than those from separator B. The unevaporated distillates flow from separator C to separator S, which is heated by the hot residuum passing through it on its way to the exchanger.

The vapors from separator S pass off to the condenser V in the same manner as those from B and C; they are heavier than those from the other two separators. The unevaporated distillate from separator S is not further refined and is run to storage.

Each of the three separators has two separate vapor chambers, each having a separate outlet connected to its independent condenser so that the three separators will give six different cuts of varying gravity of re-run products. An additional product will be the still bottoms drawn off from the separator S.

The Shell Co. is designing an additional Trumbull plant of equal capacity to be erected alongside of the present one, in which still further improvements are contemplated.

ASSOCIATED OIL CO.

Plant at Gaviota, Santa Barbara County

The Associated Oil Co. has a topping plant of about 5,000 bbl. daily capacity at Gaviota, Santa Barbara County, which consists of two systems; one a battery of ordinary stills and the other a battery of pipe retorts. As I have not dealt in this paper with ordinary stills, but have confined myself entirely to the other types, and as the Associated Oil Co. is building a much larger pipe retort topping plant at its Avon refinery, of the same general design as the one at Gaviota, I will not describe either of the Gaviota systems but will describe the topping plant now being constructed at the Associated Oil Co.'s new refinery at Avon.

Plant at Avon, Contra Costa County

A retort topping plant is being erected as an addition to the existing refinery and consists of a flat square construction, divided into three separate furnaces. (Figs. 22 and 23.) The direction of all flame is at right angles to the pipes. Placed above the three furnaces will be three rows each of 34 4-in. pipes, about 20 ft. in length with flanged return bends at either end. At the rear end of the furnaces is an enlarged flue which contains 40 4-in. pipes for the purpose of absorbing as much heat from the flue gas as is practicable. By constructing three long furnaces, over which the pipes are placed at right angles, the pipes can be supported at four points. Also, short cast-iron T-bars are placed between each row of pipes over the division furnace walls, which support the

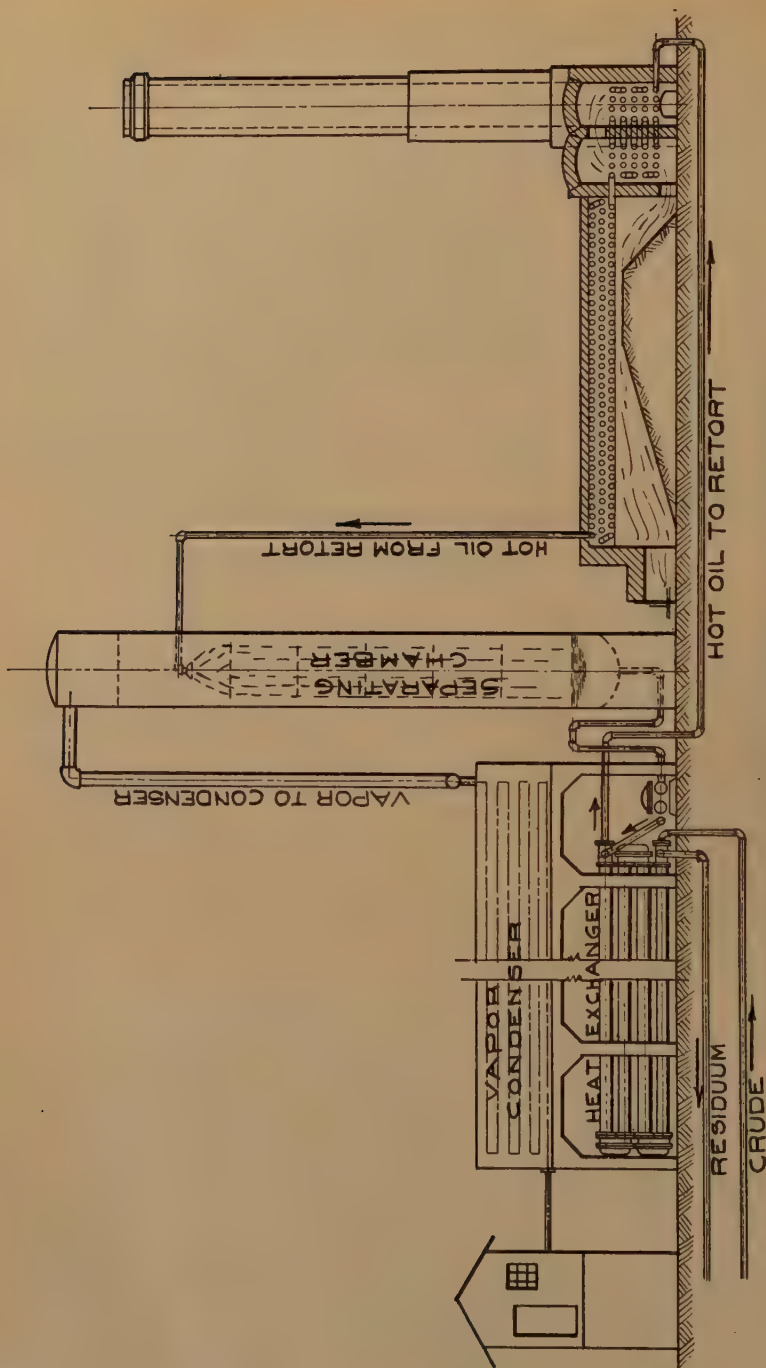


FIG. 22.—ASSOCIATED OIL CO. TOPPING PLANT, AVON REFINERY.

upper rows and prevent bending of the pipe due to the action of heat. The roof over the furnaces, which is covered by a double thickness of hollow tile with a layer of gravel, rests on the upper row of pipes.

The main outside walls of the furnace come just inside of the return bends at either end of the pipe. There is also a light auxiliary wall placed on the outside of the return bends. The space between the main wall and this auxiliary wall is covered on top with short light sheet-iron plates which prevent a circulation of cooling air on the return bends. The whole construction admits of removing any pipe or joint without loss of time.

In connection with the retort is a separating tower about 5 ft. in diameter and 40 ft. high, into which the heated residuum enters at a point about



FIG. 23.—VIEW SHOWING FRONT OF FURNACES UNDER CONSTRUCTION.

10 ft. from the top. The residuum strikes a succession of baffles as it falls to the bottom of the separator; the light vapors flow upward and out of the vent pipe at the top of the separator.

The heat exchanger consists of a nest of 24 10-in. pipes about 40 ft. long mounted on a steel frame, the 10-in. pipes being fitted with return bends, and having five 2-in. pipes passing through each from end to end with connections so arranged at either end of the 10-in. pipes that the heated residuum begins its flow at one end of the 10-in. pipes and flows backward and forward through each successive 10-in. pipe until it has reached the end; in the opposite direction the cold crude oil is admitted into the 2-in. pipes at the cool end of the exchanger and leaves the exchanger at its hottest end.

An ordinary water-cooled condenser for condensing the vapors from the separator is used in connection with the retort.

In operating the plant, the crude oil enters the exchanger, passes out of the hottest end and enters the retort, at a pressure of about 50 lb., through the lower corner pipe of the flue chamber, traverses backward and forward through the eight lower pipes in this chamber, then flows successively through each row upward to the top; from there it traverses backward and forward over the bottom row of pipes lying over the furnace until it reaches the front, then in the same manner through the second and third rows back to the front of the furnace; from there it is led, by a pipe with a pressure regulating valve in it, to the upper portion of the separator where the spraying takes place. The residuum is then pumped from the bottom of the separator through the heat exchanger to the storage tanks. The vapors from the separator are led direct to the condenser.

The residuum will be delivered from the retorts to the separator at about 450° F. As this particular topping plant has not yet been completed, I cannot give definite data as to its operation, but the same design of exchanger used in connection with ordinary stills topping oil in the same plant, receives the residuum at 450° F.; the crude oil enters the exchanger at 65°, leaves it at 260°, and enters the stills at approximately that temperature.

The estimated economical capacity of this plant with clean oil will be about 5,000 bbl. of crude each 24 hr., taking off about 15 per cent. of 50° Bé. tops.

In closing this paper, I wish to express my appreciation of the courtesies extended to me by the officers and employees of all the companies herein referred to, as it would have been impossible to compile so many authentic data without their hearty cooperation.

DISCUSSION

WILLIAM A. WILLIAMS, San Francisco, Cal.—Is the Union Avila plant run as a topping plant or as a dehydrating plant?

A. F. L. BELL, San Francisco, Cal.—Both. It takes off an average of 20 per cent. water besides topping the oil. Part of the crude oil is run through an electrical separator before running to the topping plant, but the entire crude averages about 20 per cent. when it enters the topping plant.

DAVID T. DAY, Washington, D. C.—I suppose most of the work contemplated in these large topping plants is simply to deal with topping the oils with only a small percentage of water in them.

A. F. L. BELL.—Not more than 2 per cent. of water can be handled in ordinary stills without causing trouble. At our Gaviota refinery, where

the oil runs from 15 to 30 per cent. water, we carry 20 lb. pressure on our stills to hold the foaming down, we can do that with small stills, with large ones it would not be safe; but with a retort, the quantity of water contained in the oil does not affect its working. It simply means additional fuel to evaporate the water.

DAVID T. DAY.—In your judgment is it the proper thing to dehydrate and top the oil at one operation; or do you think it would be better to dehydrate with the electrical appliances first and then top afterward?

A. F. L. BELL.—Where the water in the crude exceeded 22 per cent. I would recommend dehydrating first by the electrical, or some other process that did not necessitate evaporating the water.

DAVID T. DAY.—Would that apply to oils mixed with fresh water as well as oils mixed with salt water?

A. F. L. BELL.—Yes, with any water. The cost of fuel for refining increases in proportion to the quantity of water in the oil. The water should be separated, if possible, before it goes to the stills; but with less than 2 per cent. water such separation would not pay.

The Cost of Maintaining Production in California Oil Fields

BY M. E. LOMBARDI,* B. S., BERKELEY, CAL.

(San Francisco Meeting, September, 1915)

THE cost of maintaining the production of an operating oil company is one of the most important, as well as one of the most difficult to estimate, of the various items which go to make up the total cost of producing oil. In the opinion of many operators the cost of drilling and developing new wells, whose production takes the place of the loss in production through decreased yield of the old wells, is considered an operating expense, for the reason that the total income-producing power of the property is not increased. The cost of these new wells may be added directly to production expense, or it may be charged to capital account, and this account depreciated to the same extent. In either case, after an oil company is on a satisfactory income-producing basis, the money for drilling new wells to maintain production must be taken from the income of the company.

Three factors determine the cost of maintaining production: First, the rate of decline in yield of the wells; second, the cost of drilling and equipping new wells in terms of their production; third, the decrease in initial production of new wells as the property is drilled and gas pressure taken off the sands, etc.

In order to check theoretical figures for the cost of maintaining production the writer, with the assistance of Reed Bush and Fred Tough, has compiled the following figures. It is necessary first of all to choose a period of time during which production for the field or territory studied remains fairly constant. The longer the period of time over which this condition prevails, the better. It is also necessary to eliminate or allow for any extraneous condition which may have affected the production returns, such as the shutting down of a large number of wells, the bringing in of a well of exceptionally large yield, the opening up of virgin territory where the flush yield is greater than will be had from new wells in after years, etc.

All these conditions are exemplified most simply in the case of the Coalinga field, and perhaps these figures are more truly prophetic than the two other cases given. In applying these figures to the future it is well to assume that the average yield of new wells will constantly de-

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crease, since the flush-production period is past for most of the territory. On the other hand, the cost of drilling will be constantly less, owing to improved methods and particularly to the increasing use of the rotary drill. Although a great many estimations must be made, the results check closely enough to be interesting and possibly of value.

The data used are from the records of the Producers' Agency, the *Standard Oil Bulletin*, and various private sources.

It is to be noted that the figures for number of wells at the beginning of the periods include producing wells only, but that all wells drilled during the period, whether profitable or not, are included in the calculations, provided they are within the accepted limits of the field—not pure wild cats.

Coalinga Field

In the Coalinga field the period from Aug. 1, 1912 to July 31, 1913 was chosen. The production curve is shown in Fig. 1. It will be noted that

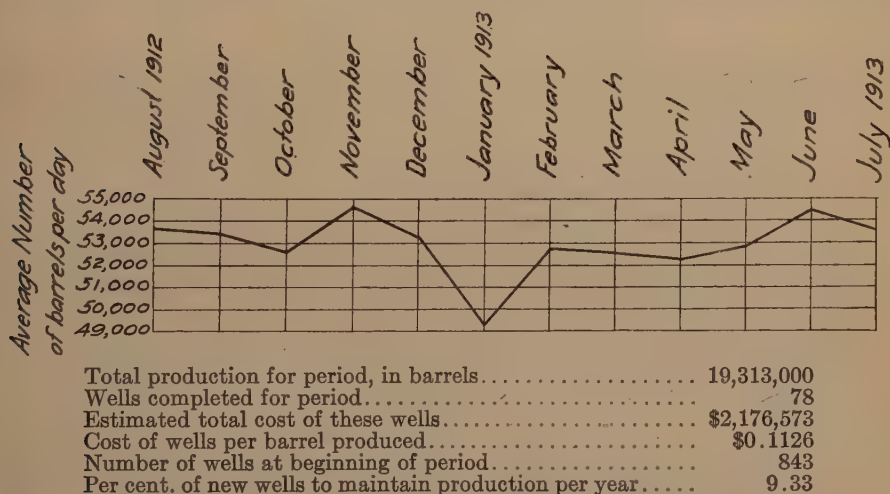


FIG. 1.—CURVE OF TOTAL PRODUCTION OF CRUDE OIL IN COALINGA FIELD, CALIFORNIA, FOR THE PERIOD AUGUST, 1912, TO AUGUST, 1913—12 MONTHS.

the production during these 12 months was remarkably uniform, the greatest variation from the average being about 8 per cent. and the average variation on a monthly basis being about 2.8 per cent. During this period 19,313,000 bbl. of oil were produced and 78 new wells were completed. The number of wells suspended during the period varied from 176 to 214 out of a total of 843 wells at the beginning. These wells were "off" from natural causes, and were not to any extent intentionally shut in. The greatest variation from the average number of wells suspended was only 8.6 per cent., so that this factor of wells shut in is not of great importance in the results.

The main facts are, that at the beginning of this period 843 wells were producing, and that, in order to maintain the production of these wells, 78 new wells had to be drilled. To arrive at the cost of these 78 wells, representing the cost of maintaining production, the new wells were classified in four groups. The actual cost of 106 wells in this field (also divided into groups of similar characteristics) was accurately known to the writer. These average cost figures applied to the 78 wells in question showed their cost to be approximately \$2,176,573. This cost includes derrick and rig, well, flow-tanks and flumes, gas engine or other local pumping equipment, but not such items as storage facilities, pipe lines, main power plants, etc. If this figure is divided by the total number of barrels produced during the period, we find that 11.26 c. per barrel of oil produced was the cost of maintaining production in this field.

It is also interesting to note that the number of wells necessary to maintain production for one year was in this case 9.23 per cent. of the total number of wells producing at the beginning of the period.

Sunset-Midway Field

In the Sunset-Midway field the period from November, 1913 through November, 1914, was chosen. Fig. 2 shows that the production was

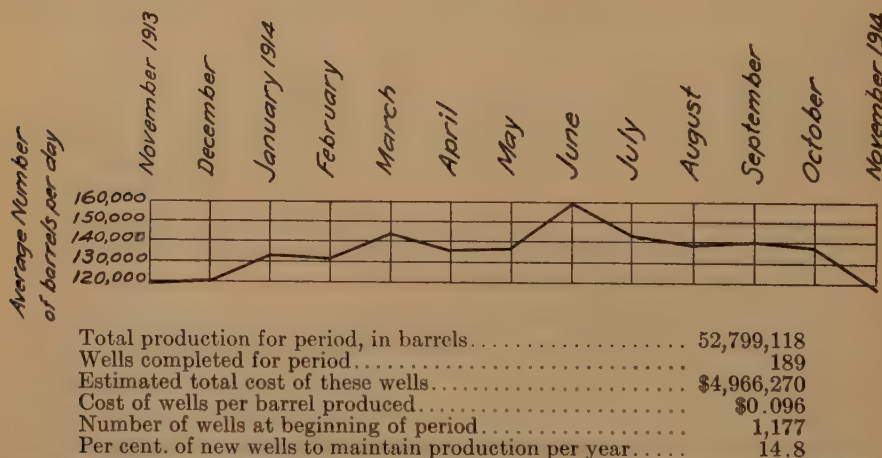


FIG. 2.—CURVE OF TOTAL PRODUCTION OF CRUDE OIL IN SUNSET-MIDWAY FIELD, CALIFORNIA, FOR THE PERIOD NOVEMBER, 1913, TO DECEMBER, 1914—13 MONTHS.

more than maintained, that is, that it was higher at all times during the period than at either end. The Lakeview No. 2 gusher came in in May, 1914 with a production of about 25,000 bbl. However, the greatest variation of the average production was not more than 10.5 per cent. During this period seven gas wells were drilled. These are not included in the figures given, as it is assumed that they were drilled to produce gas only, and no attempt made to produce oil.

During this period, a wide-spread agitation occurred in favor of limiting production, and a good many wells were closed in. The probable production of these closed-in wells was estimated as closely as possible and added to the actual production of the field, in order that this intentional decrease should not affect the results, which are of necessity based on normal flow from each well.

It will be noted that the total oil produced was 52,799,118 bbl. including estimated shut-in production, and that 189 wells were drilled during the period. The cost of these 189 new wells was estimated in the same manner as for the Coalinga field. The costs of 106 wells in various parts of the Sunset-Midway field, being accurately known to the writer, were applied to the 189 wells with due regard to the location of each well, etc., the same items being included as in the case of Coalinga. The resulting estimate is \$4,966,270 for maintaining production during the period, which amounts to 9.6 c. per barrel of oil produced.

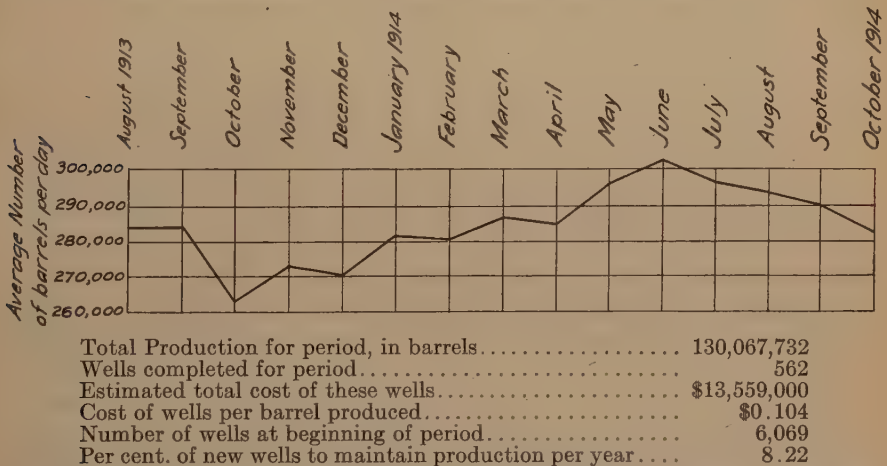


FIG. 3.—CURVE OF TOTAL PRODUCTION OF CRUDE OIL IN CALIFORNIA, FOR THE PERIOD AUGUST, 1913, TO NOVEMBER, 1914—15 MONTHS.

The proportion of new wells necessary to maintain production for one year in this case is 14.8 per cent., which is a considerably higher figure than for Coalinga, although the cost per barrel produced is less. This is explained to some extent by the fact that a larger number of new wells in the Sunset-Midway field were drilled in shallow and "easy" territory in the northwestern part of the field, with a correspondingly low cost per well.

All California Fields

A study was next made based on the production of the whole State of California for a period of 15 months beginning August, 1913. During this period, as shown in Fig. 3, the greatest variation from the average

production was only 7.5 per cent., which is remarkably small considering the period.

At the beginning, approximately 6,069 wells were producing and during the period approximately 562 wells were brought in, costing approximately \$13,559,000. This figure for the cost of new wells is not as accurate as desired, because the writer is not familiar with the cost of wells in some of the smaller fields. Average cost figures taken from over 250 wells in four fields were used. During the period approximately 130,067,730 bbl. of oil were produced, and dividing the cost of the new wells by this production, we find that the cost of maintaining production was 10.4 c. per barrel of oil produced. This differs from the Coalinga figure by only 0.86 c. per barrel, and from the Sunset-Midway figure by 0.8 c. and is very close to a mean between the two. It is also interesting to note that the number of wells necessary to maintain production over the whole State for one year was 8.22 per cent. of the wells producing at the beginning of the year, which compares favorably with the figure of 9.23 per cent. for Coalinga.

Cost per Barrel of Daily Production

It is also interesting to apply the data given above to estimate the cost of maintaining production in terms of the average daily production. For example: A company operating average leases in various fields, having a daily production of several thousand barrels, may wish to estimate from a general average how much money will have to be appropriated per year for new drilling to maintain its production.

In the following calculations the average daily production for the whole period is assumed as the production at the beginning of the period. In Coalinga the daily production may be thus assumed as 52,912 bbl. at the beginning of the period illustrated above. It, therefore, cost \$40.38 per barrel of daily production to maintain production during the ensuing year.

Likewise for the Sunset-Midway field, the figure is \$33.70, and, for all fields in California, \$38.

It must be borne in mind that, in this case, as in the case of the cost per barrel produced, these figures show the cost for wells and their immediate appurtenances only: The cost of extension of pipe lines, new permanent power plants, camps, water systems, etc., must be added. A fair figure for these is between 12 and 20 per cent. of the cost of wells.

Depreciation

If we apply the cost figures used above to all wells producing at the beginning of the various periods shown, the apparent depreciation in earning power of investment during the ensuing year is shown by the

ratio of new wells to old. In Coalinga it is 9.23 per cent., in the Sunset-Midway field 14.8 per cent., and for all California fields 8.22 per cent.

It must be borne in mind that these figures apply only when considering wells producing at the beginning of the period. That is, no account in investment is taken of past failures or abandoned wells. Also that the figures show an average depreciation for wells in all stages of life. It is well known that new wells fall off in production as much as 50 per cent. in their first year, whereas wells 5 or 6 years old decrease very slightly in comparison.

It must also be borne in mind that many of the new wells which maintained the production during the period illustrated were in practically virgin territory in the extension of the known fields, or in new fields, and that their initial production was correspondingly high. This condition will not exist long, and it will be safe to discount the results given above for future use.

DISCUSSION

C. D. KEEN, Shreveport, La.—Did Mr. Lombardi include in his figures the purchase price of the property?

M. E. LOMBARDI, Berkeley, Cal.—No.

C. D. KEEN.—Suppose I pay \$1,000 per acre for it, and in four or five years it is worked out.

M. E. LOMBARDI.—That is a question of depreciation and must be added to my figures.

MARK L. REQUA, San Francisco, Cal.—You say it cost \$35,000 to \$40,000 a year to maintain 1,000-bbl. per day production?

M. E. LOMBARDI.—Yes, 1,000 bbl. a day would cost \$40,000 a year to maintain.

M. L. REQUA.—It is rather a coincidence that the figures in my mind for some time check almost exactly with those Mr. Lombardi has arrived at, after examining the data which he has had at his command. I think he has made a conservative estimate. It is peculiarly opportune, as the average Californian has not considered what it costs to produce oil in this State. Interest on the investment, depreciation and amortization, and such matters as that have never been taken into consideration. The popular notion is that it cost so much to pump oil from the wells. The indirect costs are more than the costs of operating the wells and pumping the oil out of the ground, and that is the reason the industry is depressed at the present time. Taking the industry over the State, nobody is making money, and that is because it has not been realized

what it costs to produce oil. With oil selling at 35c. to 45c. a barrel, there is no profit for the producer.

H. M. BACON, San Francisco, Cal.—I checked that up on one of the leases under my charge running over a period of three years. Upon completion of our second well the development cost of oil produced was 67½c. The average price of all the oil sold at that time was about 57½c. As the development was continued and more wells were brought in, these figures about balanced, but with oil selling at the price obtaining today, considering operating and interest charges, the investment shows little or no profit to date.

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Protecting California Oil Fields from Damage by Infiltrating Water

BY R. P. MCLAUGHLIN,* SAN FRANCISCO, CAL.

(San Francisco Meeting, September, 1915)

IN most branches of the mining industry it is a well-recognized fact that care must be taken to protect the mineral deposit from undue physical injury. It is comparatively easy to grasp this idea when the mineral is a solid and the lives of workmen depend directly upon the care with which it is removed from its natural position, but when the product is a liquid drawn up from pools several thousand feet beneath the surface of the ground through openings only a few inches in diameter, it requires some imagination to picture underground conditions and the changes which must often occur. The development of the petroleum deposits of California has been attended by numerous accidents, the results of which are readily noticed at the top of the wells and in the profit and loss accounts of the operators. Determination of the cause of these accidents has been the subject of considerable study, which naturally has fallen upon men familiar with technical methods.

The greatest damage to the oil deposits of California has resulted from water finding its way into the oil-bearing sands, and it is the purpose of this paper to set forth briefly this phase of the oil industry. Much of the material here presented has previously appeared in publications issued by the California State Mining Bureau, which, under the direction of State Mineralogist Fletcher McN. Hamilton, has investigated the subject and taken steps toward improvement of conditions. The writer, having been in charge of this branch of the Bureau work, has gained considerable knowledge of existing conditions.

Generally speaking, the oil-bearing formations of California are soft, unconsolidated sand beds, separated by clay strata, which are also soft. Some of the sand beds carry water instead of oil. All of the beds are more or less separated, probably being of lenticular shape. Very few of the beds in the oil fields lie flat or undisturbed over any great area. Folding and tilting are common, and faulting also occurs in some of the fields. The accumulation of oil is not confined to any single structural feature, such as the anticline, but occurs in many other structural features. Water-bearing strata occur in practically every oil field in the State. Complete data are not at hand for a comparison of the damage

* State Oil and Gas Supervisor.

done in the various fields. These water sands are sometimes above, below, and between the oil sands. When water and oil sands are both penetrated by a well, the water enters the oil sand. The water frequently forces the oil back into the sand so that pumping produces nothing but water. The water not only damages the well which it first enters, but affects neighboring wells, causing their product to become partly or entirely water. The probability of such damage is usually foreseen. The method of preventing damage from this source is to insert screw pipe (or casing) into the well. It serves as the lining or wall of the well, and if carried through and below the water sand and firmly landed in some sort of impervious stratum, such as clay, it prevents the water from leaking down into the well. Sometimes it is necessary to pump cement in behind the casing in order to seal thoroughly the passages down which the water might otherwise run. Several such lines (or strings) of casing are sometimes placed inside the first one until together they resemble a telescope with the small end at the bottom of the well, in the oil sand.

The protection of a well from water intrusion comprises two operations: First, determining where the casing should be landed; and, second, the actual work of placing the casing in the well in a workmanlike manner. These operations are interdependent, and either is useless without the other.

Some idea of the financial loss of the oil operator, when water is not controlled, may be had from the following instances:

In the Coalinga field on a certain property is a well about 2,700 ft. deep. It was drilled three or four years ago at a cost of about \$30,000. Initial production was over 200 bbl. of oil per day and the total production about 100,000 bbl. of oil, giving a profit of possibly \$10,000 over regular operating expenses, but not including the cost of the well. About two years ago water stopped this production entirely. The owners are financially unable to repair the well at present, and each day adds to the damage being done to surrounding property.

In a portion of the Midway-Sunset field there are two known oil sands, both productive. There are water-bearing sands above the upper oil sand and also between the two oil sands. When a well is drilled into the lower sand two extra strings of casing (about 2,000 ft. of 8-in. and 2,500 ft. of 6-in.) should be inserted to keep the water out of the upper sand. The expense for extra casing and labor may amount to \$10,000 per well. Some operators have neglected or refused to protect their wells in this manner, which has resulted in serious damage to the wells of other operators which depend upon the upper sand, because they are of too small diameter to allow deeper drilling.

An approximate idea of the ultimate loss that will ensue, unless the top sand is protected, may be gained from the following figures: A

single well, at present producing, has during three and a half years produced almost 300,000 bbl. of oil. Other wells, though not so large, should produce for a much longer time than three and a half years, and 300,000 bbl. each is not an unreasonable estimate of their total production. The upper sand extends over about 400 acres, and at 5 acres per well should supply 80 wells, or a total of not less than 24,000,000 bbl., or \$12,000,000.

A minor instance which came under notice is typical of much inefficient work that has been done. A superintendent, who had previously finished several wells satisfactorily with advice from a geologist, proceeded without such advice and drilled another well too far through the oil sand and encountered a strong flow of water. This "bottom water" was encountered within 2 ft. of where it had been expected by the geologist, from his study of logs of surrounding wells. Advice would have been to stop drilling about 15 ft. above the point to which it was carried. A contractor was engaged to cement the bottom of the hole after the flow of water had been checked by a "bridge." Even if the first cementing operation proved successful, the needless expense on this well probably amounted to about \$500. The chief aim of the geologist or engineer is to meet such circumstances *in time to prevent trouble*, rather than to aid in patching up improperly drilled wells, which procedure is, at best, apt to be troublesome and unsatisfactory.

It must be borne in mind, of course, that the natural oil reservoirs are exhaustible and as the oil is removed water is apt to take its place. Some operators are prone to assume that the presence of water in an oil well is perfectly natural and that infiltration could not have been avoided. It has been proved that this hypothesis is not always correct, for in many cases where water has appeared, the correction of faulty material or workmanship has caused the water to disappear.

There are at least three causes for water entering oil wells: (1) failure of casing, (2) faulty drilling operation, and, (3) natural replacement of oil by water as the oil is pumped out. There are also combinations and variations of these causes, modified and accelerated by movements of the ground itself.

The accompanying sketch (Fig. 1) of the bottom of four wells in the Coalinga field illustrates some of the underground conditions and the methods of recording and presenting the data. The wells were drilled in order as numbered. The strata are not bent or deformed in this particular locality. In each well the 10-in. casing was intended to shut off the water occurring above the oil sands, and in well No. 3 it was landed at a depth of 1,880 ft., in conformity with successful results at No. 1 and No. 2. However, the sand between 1,980 ft. and 2,016 ft. was found to contain some oil and considerable water, hence it was necessary to use more casing to shut off this water. Since some experimenting was

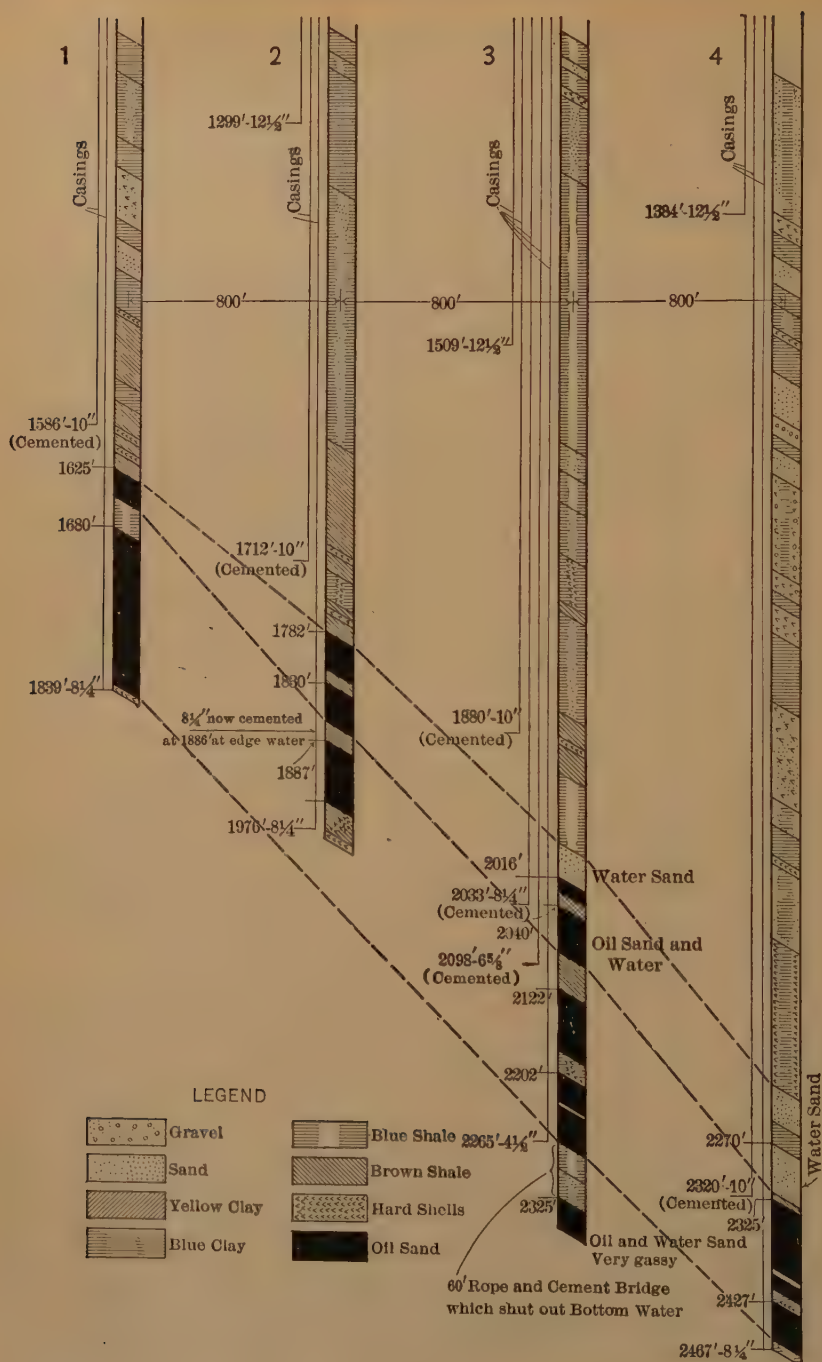


FIG. 1.—CROSS-SECTION OF LOWER PART OF FOUR WELLS IN THE COALINGA OIL FIELD, CALIFORNIA.

necessary, two strings, the $8\frac{1}{4}$ in. at 2,033 ft. and the $6\frac{5}{8}$ in. at 2,098 ft., were used. Had all conditions been known beforehand only one string would have been necessary, but when the $8\frac{1}{4}$ -in. casing was landed and the well drilled deeper more water was encountered and the $6\frac{5}{8}$ -in. casing had to be used. The final oil string of casing, $4\frac{1}{2}$ in. in diameter, was carried to 2,351 ft. and upon testing, the well was found to produce mostly water, so the bottom was plugged (using waste, rope, iron lathe cuttings and cement) up to 2,265 ft. and the well again tested, when it was found to be free from water.

Since well No. 3 was one of the first wells in this locality to encounter water in the oil sands, where oil was expected, careful work was necessary on the next well drilled, to check the operations. This was done on neighboring wells, not shown on the drawing, and since similar results were found, when well No. 4 was drilled the top sands were shut off with the 10-in. casing at 2,320 ft. The well was drilled to 2,467 ft. only, so as not to encounter the sand which contained water in well No. 3 between the 2,325 and 2,351 ft. levels.

After completion, all these wells produced clean oil for about a year, and then the No. 2 began to show a little water, and about three months later showed about 80 per cent. There were three possible sources of the water: From other wells through the oil sand, or from above the oil sand, or from the bottom sands penetrated. The drawing shows that the possibility of bottom water was slight. The possibility of water from above was tested by removing the $8\frac{1}{4}$ -in. "oil string," putting a "bridge" or plug in the open hole between the bottom of the 10-in. "water string" and the top of the oil sand. After bailing the well dry above the plug no water came in, proving that the "water string" was not leaking at any point and was effectively excluding upper water. Therefore the water in No. 2 was probably coming through the oil sand itself, and as the same sand showed both oil and water between 1,980 ft. and 2,095 ft. in No. 3 it was concluded that the water was rising along the dip of that sand as the oil was removed. Such a movement is called the encroachment of "edge water." The $8\frac{1}{4}$ -in. casing was again put in and cemented at 1,886 ft. in the shale presumed to correspond to that found at 2,091 to 2,122 ft. in well No. 3, and 2,318 to 2,325 ft. in well No. 4. After pumping the well for about 10 days, the oil was found to contain less than 1 per cent. water, proving that water had been coming through the upper sands, between 1,782 and 1,870 ft. After about four years the water has commenced to show in well No. 1. This movement naturally follows the removal of oil and cannot be stopped, but the particular sand affected can be kept separated from other productive sands.

As in all other important industries involving the winning of some mineral product from the earth, a thorough plan should be mapped out, and followed or revised in the light of new developments. In other words,

there should be a thorough geologic investigation. Some large operators realize this and have facilities for thorough geologic work at all times. Other large concerns give but little attention to the subject. Probably the oil industry is in this respect not so well managed as the metal-mining industry. Small operators frequently feel that they cannot afford the expense of geologic work.

Several hundred oil companies are operating in California and in the past no concerted effort has been made by them to develop their lands systematically or to control the water trouble. Constant agitation of the subject has resulted in the passage of a law (Chapter 718, Statutes of 1915) framed by the State Mining Bureau, at the invitation and with the assistance of many oil operators. The law became effective Aug. 8, 1915. As it is along original lines, a brief outline may be interesting, together with some remarks upon the aims of its sponsors.

The law gives the State Mining Bureau supervision over this work, the expense being met by assessments upon the producers. The work is directed by an officer designated as State Oil and Gas Supervisor, who appoints four deputies; all of these officials must be either engineers or geologists. The Supervisor's headquarters will be at the State Mining Bureau in San Francisco and the deputies will reside in the various oil fields. Operators are required to file all records of work done on wells and other pertinent information. The records are to be considered as confidential unless released by the operator furnishing them. The Supervisor is empowered to order repair work done on wells and if it is not done he can place agents on the ground to do the work. The cost of the repair work becomes a lien upon the property. Abuse of power by the Supervisor is prevented by making his orders subject to review and reversal by an arbitration board appointed by operators.

The law is rather lengthy and explicit, but no rigid construction can be expected to solve such varied problems as exist in our various oil fields. It is believed that operators generally will cooperate with the officials in the endeavor to make the law fit economic and physical conditions.

The underlying purpose of the framers of the law was to establish a technical department that would gather all the facts and present conclusions to those having charge of drilling operations. In other words, it is hoped that the department will fill the place of a consulting engineer. The need of such a position has been recognized by some operators in their own private organization, while others have not realized its value or have been unable to bear its expense. It is therefore evident that the first work of the State Mining Bureau must be educative. Every man engaged in the work must primarily be able to touch the chord of human interest and present the technical facts to his audience, the operators, in such a fashion that they can see the subject as clearly as he does. Technical workers have frequently complained that their work is not given

proper recognition by the public, and in many cases this has been due to the fact that no effort was made to grasp the thread of public thought on that special subject and turn it into the proper channels. The legislation here considered presents one of the best opportunities for those administering it to demonstrate whether or not technical men are able to take hold of a large problem and present it in intelligible form to men not technically educated but thoroughly skilled in drilling deep and difficult wells.

Not alone to the oil operators, but also to the public which is interested in conservation of the oil, as well as all other natural resources, is the Bureau under obligation to solve the problem outlined in this paper. California drillers have already proved their ability to cope successfully with the most severe physical conditions and will undoubtedly be found willing to assist the engineer who now has an opportunity to demonstrate his usefulness.

DISCUSSION

M. E. LOMBARDI, Berkeley, Cal.—Mr. Bell, I would ask you to state before the meeting what you said to me the other day, about the final recovery of oil from oil sands that have become inundated with water.

A. F. L. BELL, San Francisco, Cal.—In stating my opinion, I give it with certain qualifications. In the Kern field, and possibly in some others; I think we will eventually get all the oil out, whether the water comes in or not, since it is peculiarly constructed, being in the form of a partial dome. In 1909, while I was field manager of the Associated Oil Co., I installed some large air compressors to handle the water. At that time the oil had migrated to the higher and more shallow portions of the field, because the water had inundated the deeper sections. The result was that the shallow districts within the center of the dome were in many cases producing as much or more oil per well than the deeper wells. And in the inundated sections the water had slowly spread over large areas, so that nothing but water was pumped from them. Under these conditions, we decided to pump the wet well with air, and installed four 2,000-cu. ft. compressors.

The air was compressed and forced into the wet wells at about 250 lb. pressure. Shortly the inundated district that had produced almost all water and only about 150 bbl. of oil a day increased its production to over 2,000 bbl. of oil per day. It was not long before we noticed that the production of the shallow territory was falling off to such an extent that in many cases the wells were abandoned; at the same time the area of the water-inundated district was greatly reduced, showing that when the water pressure was reduced the oil migrated back to the lower depth.

The wells which at first had had the air jets put into them finally produced so much oil that we had to take the air jets out and return to the use of the beam pump, because it is not commercially feasible to pump an oil of 14° gravity with air. To handle such oil profitably by air, it must carry with it 50 to 90 per cent. of water.

The results have led me to believe that on monoclines, where there is a gradual rise to a certain point, or in plunging anticlines, even though water gets into oil sands, eventually we get the larger percentage of the oil. However, the man with a small acreage in the deeper sections may find some morning that his oil has gone, another man in some higher portion of the district having gotten it.

I do not use this as an argument against any of the means adopted for the prevention of the inundation of water, since infiltration is always a source of danger. Many of my theories have not always worked out, and this is only a theory of mine.

A. C. McLAUGHLIN, San Francisco, Cal.—Supplementing what Mr. Bell has said, the division ownership of lands makes necessary some solution of the water question. Mr. Bell's theory is undoubtedly correct in some cases. However, if there are some 25 or 30 owners, each would like to get the original amount of oil which underlaid his lands.

A. F. L. BELL.—My theory is that to get oil out of an inundated district, you must reduce the liquid level so that water and oil flow together to the well; but if the water level cannot be reduced and remains at a high level in the well, then the water is a detriment and will shut off the oil. To get all the oil you must have the water under such control that you can pump more water and oil from the well than is coming to it. The moment more water comes to a well than can be handled, then the oil is shut off.

A. C. McLAUGHLIN.—By reason of the higher surface tension of water over oil, the tendency is for the water to migrate to lower pressure areas; where pressures are practically nil, the conditions are as Mr. Bell mentions them, but in cases where there are pressures of 25 or 30 lb., the tendency is to migrate toward the well. This displaces the oil and creates an area of low pressure when a well is dug. So I think it is conceivable that by letting water into oil wells, the oil could never be gotten out.

WILLIAM A. WILLIAMS, San Francisco, Cal.—The Bureau of Mines is charged with the supervision of operations on Indian lands in Oklahoma, and we have had the water menace to contend with on these lands. The Cushing field is perhaps one of the best illustrations of the damage that may be done to a field by not effectively excluding the

water from the oil and gas sands. On March 1, 1915, the production of the Cushing field was in excess of 300,000 bbl. per day; today it is less than 140,000 bbl. The excessive decline is due to improper methods of casing wells to shut off the water. The operators did not case off the water uniformly; some cased above a certain sand and some cased below the same sand. As there are several producing sands, both oil and gas, in the Cushing field, the result of this unsystematic casing has been intercommunication between sands, permitting the gas to come into contact with the water-bearing sand, and while the gas pressure was sufficient to overcome the hydrostatic pressure of the water sands, the gas forced the water away from the bore hole. When this gas pressure was not sufficient to overcome the hydrostatic pressure, the water found its way into the oil sands, resulting in an abnormal decline in production and the formation of a considerable percentage of cut oil or emulsion. By the drowning and wasting of gas a considerable loss occurred not only in the value of gas as a fuel, but in the matter of operating costs, for where gas occurs with the oil, this gas, when properly conserved, assists in pumping the wells, thus reducing operating costs.

It was found impracticable to protect with a string of casing each sand found in the Cushing field and in lieu of the large amount of casing necessary to protect the sand adequately, the Bureau of Mines has recommended sealing with mud all oil, gas, water, and barren sands, above the productive sands. This is accomplished by landing a string of casing immediately above the sand to be utilized, sealing with mud all sands behind this casing.

In one of the Kansas fields, where the gas occurred at 1,500 ft. and oil at 2,500 ft., a strong flow of top water was encountered above the gas sand. Wells were drilled, the water and gas sands mudded off behind a single string of casing; the oil wells were drilled in at a cost \$500 less than required to complete wells without the use of the mud fluid, notwithstanding that the well had to be drilled for a considerable distance with the fluid in the hole. The saving was accomplished in the reduced amount of casing required, the company drilling separate wells for gas.

A. F. L. BELL.—A law should be enacted requiring that all sands above the oil-producing sand should be cased off.

WILLIAM A. WILLIAMS.—It is necessary that all water sands, in fact all sands, be cased off uniformly. Unless this is done, water will find its way through the sands into the producing sand notwithstanding the fact that water may have been effectively shut off at different casing points in the various wells.

MARK L. REQUA, San Francisco, Cal.—I would like to ask Mr. McLaughlin whether he has decided how far he can go in remedying the

water situation. I believe the conditions in certain parts of the State have gone to such an extent that it is dubious whether they can be remedied.

R. P. McLAUGHLIN.—Many old wells have been drilled with no records to show what was put in them and what the conditions were originally. Many of those old wells we cannot remedy since it might cost more to fix them than the wells are worth.

This would be a mere guess, but out of all the old wells we would be fortunate if we could at present fix up 10 or 15 per cent. However, in say, 90 per cent. of the new wells, we should be able to give the operators advice so that we could entirely avoid the damage. One of the largest fields in the world, the Midway-Sunset, has now only about one well to 30 acres of proved oil land, which gives an idea of the great importance of wells yet to be drilled.

MARK L. REQUA.—I am in entire accord with this work, but are we not, in a measure, shutting the door after the horse has run out of the stable. It is true, so far as new wells are concerned, that it will be possible to do much under the supervision of the State Mining Bureau, and in a large measure to avoid the unsatisfactory conditions which exist at a lot of wells already drilled, but in many old wells it will be impossible to shut off the water.

I have a case of that kind in the Coalinga field, where we drilled a pioneer well. We did not know the formation at the bottom and drilled into an underlying water sand which was not recognized at the time. The hole was not plugged; the casing collapsed, and we redrilled the well. We do not now know where the old casing is. It is within a reasonable limit of the bottom of that hole, but we are utterly unable to get down over the exact spot, clean the hole, and plug it.

That well is producing water, the old casing allowing the entrance of water from the underlying strata.

WILLIAM A. WILLIAMS.—The water might be shut off by cleaning out the well and filling up with mud, if the well is to be abandoned. In Illinois they have had trouble with edge water. When edge wells begin to show water, the operators cement up 10 or 15 ft. of the bottom of the sand, effectively shutting off the water until the water level rises above the top of the cement. The well is again cemented and so on until all of the oil has been exhausted from the sand, when the sand should be effectively plugged, otherwise water may get in and pass through the upper sands into the producing wells. The method of cementing was simple but effective, consisting of placing the necessary cement in the bottom of the well, forcing the cement back into the sand

with a pressure equal to a column of water possibly 100 ft. above the ground-water level.

C. D. KEEN, Shreveport, La.—In Louisiana we have no laws which deal with the shutting off of water from the oil sand. About two years ago the Governor of the State appointed a committee of representative oil men from all parts of the State, and requested them to formulate a set of regulations dealing with the conservation of oil and natural gas.

This committee submitted its report, which passed through the legislature, and is known now as the Conservation Law of the State of Louisiana. I was a member of that committee. The point was brought up, whether it would be necessary to embody in the regulations a section, making it compulsory for the oil operator to shut off the salt water from the oil sand.

After some discussion it was the belief of the meeting that the successful shutting off of the water from the oil-bearing strata is of such vital importance for the oil operator that it would not be necessary to make any regulations covering that subject. It is impossible in our part of the country to complete an oil well unless the water from the strata above has been shut off entirely.

It may be interesting to give you a short description of the methods we use to shut off the salt water.

In our wells we set a string of 6-in. casing at a depth ranging from 2,200 to 2,500 ft. After we have drilled the hole down to the point where we wish to set the 6-in. casing, we put the casing into the hole; when it reaches the bottom of the hole we displace the drilling mud inside the 6-in. casing by running about 500 ft. of plugged 4-in. drill pipe inside it. We then mix our cement, usually 30 sacks of cement and 15 sacks of sand, and pour the cement on the heavy mud in the 6-in. casing. When the cement is poured into the 6-in. casing we follow it up with a wooden plug about 3 ft. long and $5\frac{1}{2}$ in. in diameter, provided with rubber washers on both sides, so as to make a tight fit in the casing. After the plug is driven into the pipe, the swivel is screwed on and the string of pipe is lifted up about 8 in. from the bottom; then the pump is started, and the cement with the plug behind it goes to the bottom of the hole. The pump keeps on running until all the cement is pumped outside the bottom of the 6-in. casing, and the plug hits the bottom; this shuts down the pump, and we know that there is no more cement left inside the 6-in. casing. In order to make sure that the job has been a good one, a steel measuring line is run into the casing to see if the wooden plug is actually down on bottom.

We allow the cement 10 days to harden, and then bail the casing dry. If the cementing has not been successful, we cement the 6-in.

over again, or set a string of $4\frac{1}{2}$ -in. casing inside the 6-in., but under no circumstance is the well completed unless the water is shut off satisfactorily. The shutting off of the water in Louisiana has not given much trouble. Until about three years ago we had to rely on gumbo formations to set our casing in, but with the cement process we can set the casing in almost any formation to shut off the water.

I find that the process of cementing in California is somewhat more complicated than ours, which may be explained by the fact that the casings are bigger than ours. It is well known that it is easier to cement a string of 6-in. casing than a string of 12-in. casing, and I doubt very much if cementing with the single plug would be successful in California.

The committee mentioned above made one regulation which I would like to mention. The surface formations in Texas and Louisiana are soft and unconsolidated. In the new fields we meet with gas pressures ranging from 450 to 800 lb., and the blow-outs we have are sometimes serious. In different places blow-outs took place, forming craters in less than 24 hr., in which the derrick and all machinery were buried; one of the most notable instances of that kind was the burning gas well close to Oil City, La., where a crater was formed about 250 ft. in diameter and about 50 ft. deep. In order to deal successfully with conditions of that kind it was deemed necessary to set in each well a sufficient amount of surface casing, approximately 200 or 300 ft., and cement it. When the surface casing is cemented in good shape, and a blow-out is encountered, one has at least one good string to rely on, and the operator is in good shape to kill the blow-out.

In connection with my remark about the use of big casings I would like to ask why such large casings are used in California. A few days ago I visited the Coalinga field, and was told of a well in which was set 2,800 ft. of 12-in. casing. The well was drilled all the way down to the casing point with the rotary system. Would not 8-in. casing have served equally as well?

In our Gulf Coast fields, we get wells in which the casing that shuts off the water is not bigger than 6-in.; and we get some good wells, which make all their production through 6-in. casing and $4\frac{1}{2}$ -in. liner.

In my opinion, to use a smaller casing in California would save both time and money.

M. E. LOMBARDI.—We always try to shut off with 10 in., where the collapsing pressure which the pipe will have to bear does not require too heavy a pipe. The object is to finish with $6\frac{5}{8}$ or $6\frac{1}{4}$ in. at least, as we find a hole less than $6\frac{1}{4}$ in. an expensive one from which to produce.

We have to use such small holes, at times. In the western side of the Coalinga field we shut off with 10 in. and finish with $8\frac{1}{4}$. But a condition obtains there (I do not know whether it is edge water coming

in, or what), that often requires a second shutting off. If it is first shut off with 8 in., the second requires $6\frac{1}{4}$, and next $4\frac{1}{2}$, and that is not a proper or economical well.

A. C. McLAUGHLIN.—What percentage of the cementing operation is not successful?

C. D. KEEN.—The majority are successful. I have cemented about 45 wells, and I lost only one cementing job on account of some gas escaping.

WILLIAM A. WILLIAMS.—What kind of set shoes do you use in Louisiana?

C. D. KEEN.—We use a light shoe, about 2 ft. long. Some operators only use reinforced couplings. We do not use heavy shoes at all.

A. F. L. BELL.—Answering Mr. Keen's question, why we use large casing, I might say that in California the conditions are so different from those in the East that he is not the first or only Eastern man who has thought we were extravagant in our work; but there are conditions arising which cannot be understood or appreciated by anyone who has not operated in California fields. Many wells in the East are drilled and placed on the pump for what it costs us to build and equip the derrick and rig on our deep wells. The latter cost us about \$4,000, and it would be the height of folly to erect them for less. We have formations entirely unknown in the East, consisting of caving ground, and running sands. These are so different from the solid sandstone and limestones in the East that in our experience we have found it was a good plan to use larger pipes and more of them than would be necessary in many other districts.

C. D. KEEN.—In our company we finish all our wells with the rotary. We drill to the oil sands with thin mud, set the liner in the well, and wash it inside and outside with clear water.

In California, the opinion is that the rotary mud will hurt the well by filling up the voids in the sand. If there is the least gas pressure present to bring the water back out of the formation, it does not hurt the well at all to drill with a rotary.

A. C. McLAUGHLIN.—With rotary drills some wells in California have been drilled in which the oil sand was passed entirely through without its being recognized. The rotary is not satisfactory unless the depth of the oil sand is known before drilling begins.

A. F. L. BELL.—In California our formations do not lie as level as those in the East. In fields that I am familiar with in the East you

can safely say that the drillers are going to strike the oil sand in a few feet. In California the conditions are entirely different. There are so many underlying dips and folds that extreme care must be taken in locating the sand in many of the fields. In one case, in the Midway field, we drilled two wells with the rotary at the same time, and in each case we passed through the oil sand without knowing it, and the only thing noticeable while drilling was a few bubbles of gas at one point. In looking for the oil we drilled down into water and at first thought the district was barren, as there was not the slightest indication of oil. Then, after studying the conditions, we thought that possibly the oilsand might be where the little ebullition of gas had been noted. We then cemented off the bottom water in each well; one well came in at 10,000 bbl., and the other at 5,000 bbl. In still another case the rotary showed great quantities of oil in the mud and we expected a gusher, and got a 15-bbl. well. We decided to stop the rotary drilling above the oil sand and finish the well with standard tools.

The Possible Occurrence of Oil and Gas Fields in Washington

BY CHARLES E. WEAVER,* PH. D., SEATTLE, WASH.

(San Francisco Meeting, September, 1915)

DURING the past few years there has been considerable activity in prospecting for oil and gas in several parts of western Washington. From time to time seepages of oil or emanations of gas have been reported from various places. Several companies have been organized for the purpose of drilling, but only a part of these have actually begun operations. In certain localities very small seepages of oil occur, but up to the present time no oil or gas in commercial quantities has been obtained.

In order to determine the conditions which may be favorable or unfavorable to the occurrence of petroleum it becomes necessary to know the character and extent of the geological formations in which such deposits might occur. In other regions where producing wells have been developed it is possible to know what formations contain the oil and what structural conditions influence its concentration in reservoirs or zones. It is usually possible to determine those strata which contained the materials from which the petroleum may have been derived and also the position and distribution of the more porous sandstone belts into which it may have collected. Detailed stratigraphic surveys may even determine the depth at which a certain known oil-bearing zone may be tapped at any particular place. It is also possible to recognize those formations which are presumably barren of oil, as well as those which are absolutely so.

In new regions far distant from any of those which are now producing or have produced in the past, it becomes absolutely impossible to foretell whether oil may exist in commercial quantities or not. Assuming that such deposits do exist, it is extremely difficult to predict at what depth the producing zone or zones occur. However, in prospecting in a new and unexplored region, use should be made of such laws as have been found to govern the occurrence of petroleum deposits in other well-developed fields. Such deposits in their world-wide distribution are found almost exclusively associated with sedimentary rocks. Areas composed predominately of igneous and metamorphosed rocks are to be regarded as extremely unfavorable localities for the occurrence of commercial deposits of oil or gas. Assuming that such oil deposits are

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present in an undeveloped district, they would not necessarily be found associated with all sedimentary formations present. In order that oil or gas may occur it is necessary that former organic material be present, or at one time have been present, which could have acted as a source for such deposits. Conditions favorable for the imbedding of such organic material and the requisite action of bacteria should have existed. After a formation has been developed which could act as a source for petroleum deposits there should also exist porous belts of strata, such as sandstones, which could act as a retaining zone of saturated sands. In order to prevent the escape of deposits from such a zone, there should be some relatively impervious capping. Assuming that all the foregoing conditions have been fulfilled, such strata should have been subjected to structural movements so as to develop anticlinal and synclinal folds or a doming, in order that the specific gravity relations between the oil deposits and underground waters would allow the former to segregate beneath the higher portions of the anticlines or domes. It would further be necessary that erosion should not have proceeded far enough to cut through the impervious capping and allow the various constituents of the petroleum deposits slowly to volatilize. In discussing the possibilities for the occurrence of petroleum deposits in Washington it is necessary to inquire to what extent the above mentioned conditions obtain.

Topographically the State of Washington is usually considered as being divided into six provinces. These are: The great basaltic plateau or basin area; the Okanogan Highlands; the Cascade Mountains; the Puget Sound Basin; the Olympic Peninsula; and the Coast Ranges of southwestern Washington. The Great Basin area consists of Tertiary formations which for the most are composed of lava flows of enormous thickness and interbedded sands and clays of fluvial and lacustrine origin. It is absolutely certain that no marine sediments were deposited in this region during the Tertiary. These deposits rest upon an extensively eroded floor consisting of pre-Tertiary igneous and metamorphic formations. The Tertiary rocks have been thrown into shallow synclines and anticlines. The Tertiary lava flows may be entirely eliminated as a possible source for commercial quantities of oil or gas. The interbedded sediments contain no marine organic remains. The only fossil remains which have been seen are those of leaves and carbonized fragments of wood. It is probable, however, that a few species of lacustrine animal life existed in the Tertiary lakes. No organic materials which could have acted as a source for petroleum deposits appear to have been associated with the Tertiary formations of the Great Basin area. Because of these facts this area should be regarded as very unfavorable for the finding of commercial deposits of petroleum.

The Okanogan Highlands embrace the northeastern portions of the

State and are composed of Paleozoic, Mesozoic, and Tertiary formations. The rocks of the Paleozoic and Mesozoic consist of sandstones and shales together with intercalated lava flows, all of which have been extensively metamorphosed into quartzites, slates, schists, and greenstones. The entire series has been invaded by great batholithic masses of granite and granodiorite. In places resting unconformably upon these are lavas and lacustrine deposits of Tertiary origin. As in the case of the lavas of the Great Basin region, all of the igneous rocks in the Okanogan Highlands may be eliminated as a possible source for deposits of petroleum. The older sedimentary rocks of the Paleozoic and Mesozoic may possibly at one time have carried oil, although there is no evidence suggesting that such is the case. However, if they did, such deposits would have been destroyed by subsequent metamorphism. The Tertiary sediments are not metamorphosed but are of non-marine origin. Fossil plant remains occur, but no other organic remains are abundant which could have served as a source for commercial deposits of petroleum. From such evidence as is available concerning the geology of the Okanogan Highlands there is nothing to warrant the supposition that commercial deposits of petroleum may be present. The facts point strongly against its occurrence.

The Cascade Range extends almost due south from the British Columbia line into Oregon. The main part of the range attains elevations of from 4,000 to 6,000 ft. Resting upon the dissected surface of this mountain mass are later volcanic cones such as Mount Ranier, Baker, and Adams. The northern portion of the range is composed, as in the case of the Okanogan Highlands, of a great series of metamorphic formations of Paleozoic and Mesozoic age. Lying upon these older rocks or folded down into them are residual patches of Tertiary lavas together with lacustrine and fluviatile deposits. The surface of the old metamorphic complex decreases in elevation southward and is covered with an enormous thickness of Tertiary lavas. These have been deeply cut into by the Columbia River along the Oregon-Washington boundary line. At this point the surface of the old metamorphic complex is presumably below sea level. Throughout the entire Cascade Range, all the igneous formations may be regarded as totally barren of oil deposits. The same may be said regarding the pre-Tertiary metamorphics. It is barely possible that certain portions of these older formations may have escaped metamorphism and might possibly contain oil, although there is absolutely no evidence to suggest that such deposits are present. On the eastern slopes of the Cascade Mountains there are extensive areas of Eocene sandstones and shales containing coal seams. These sediments are presumably of lacustrine origin. No organic remains have been found within them to suggest a possible source for petroleum. From such knowledge as we have concerning

the geology of the Cascade Mountains it seems fairly safe to assume that the conditions are adverse to the formation or accumulation of petroleum deposits.

In the Puget Sound Basin, southwestern Washington, and in the Olympic Peninsula the geologic conditions are entirely different from those obtaining in the Cascades or in eastern Washington. The Tertiary formations of western Washington consist of sandstones and shales of marine, brackish-water and fresh-water origin. They were formed during the Eocene and Miocene periods, in embayments of the ocean or in estuaries not far from the shore line. The fossil remains of marine animal life occur in these strata. Intercalated with the Eocene beds of both marine and brackish-water origin are basaltic flows. All of these strata have been subjected to deformational movements and are now involved in anticlinal and synclinal folds. Later these strata were subjected to intensive erosion and many of the folds have been deeply cut into. During the Pleistocene, these rocks were heavily veneered with thick deposits of glacial drift.

The Eocene formations consist of 10,000 or 12,000 ft. of sedimentary strata and intercalated basalts. In King and Pierce Counties and other portions of the northern Puget Sound Basin these strata are almost entirely of estuarine origin. They contain numerous interbedded layers of carbonaceous seams but very rarely any deposits of animal origin. In southwestern Washington marine sediments prevail, although brackish-water bands are found interbedded. In the lower portion of the Eocene, the igneous flows prevail. However, marine and estuarine lenses are intercalated. The Miocene strata in both southwestern Washington and in the Puget Sound Basin attain a maximum thickness of 10,000 ft. Along the northern border of the Olympic Mountains the Miocene is nearly 19,000 ft. thick. It rests unconformably upon the Eocene basalts and shales, which in turn are lying unconformably upon the upturned edges of the older Mesozoic rocks of the Olympic Mountains. The Tertiary formations in the western part of the State are being prospected for possible deposits of petroleum. What are the geological conditions which might warrant the supposition that petroleum deposits are present in the Tertiary formations? The larger portion of the Tertiary formations in the Puget Sound Basin are thickly covered with glacial drift. Occasional exposures of bed-rock project up through the drift. It is from such exposures that the character and structure of the older rocks must be worked out. A prominent spur from the Cascade Mountains extends westerly into Kitsap County. The formations involved within it are of Eocene and Miocene age and have been folded into a well-defined east to west anticline. The Eocene rocks are composed of brackish-water sediments with intercalated basalts. Resting unconformably upon these are 10,000

ft. of lower Miocene shales and sandstones of marine origin. The core of this anticline has been subjected to such extensive erosion that the Miocene strata which formerly arched over the axis of the anticline have been completely removed. The truncated edges of the Miocene strata, along the entire course of the anticline, are exposed to the surface except where covered with drift. To the north a broad synclinal basin exists and the Tertiary rocks are covered with an enormous accumulation of glacial deposit. To the south of this anticlinal axis no exposures of the Miocene are known to exist until the Chehalis River is reached. The Eocene basalts cannot possibly have been a source for petroleum. Such sedimentary strata as are intercalated with the basalts are of brackish-water origin and could not have acted as a source for oil. The overlying Miocene strata contain marine fossils which might possibly at one time have acted as a source for petroleum. Assuming that such deposits may at one time have been formed, it would be impossible for them to be preserved to-day inasmuch as the strata along the axis of the anticlinal fold which might have retained them have been completely removed by erosion. If boring were undertaken in Seattle or to the north it would be necessary to penetrate an enormous thickness of drift, and when the Miocene strata were reached they would be found to exist close to the axis of a synclinal trough. The geological conditions in the Puget Sound Basin point very strongly to the fact that commercial deposits of petroleum cannot be expected to occur in this region.

From the southern portion of the Puget Sound Basin south to the Cowlitz Basin the Eocene formations have been thrown into shallow folds with axes having a prevailing trend from northwest to southeast. The marine phases of the strata contain organic remains which might possibly have acted as a source for small amounts of petroleum. No surface indications to denote the occurrence of petroleum are definitely known to exist. There is no geological information present to warrant the statement that petroleum deposits cannot occur in Thurston, western Lewis, and Cowlitz Counties. Neither are there facts to suggest the occurrence of commercial deposits. The probabilities are that if they should be found to be present it would only be in comparatively small amounts, as no extensive remains of organic material occur which could have acted as a source of origin.

The southwestern portion of the State contains Eocene and Miocene strata with associated marine organic deposits. These have been folded. It is possible that oil may be present in small quantities beneath the anticlines. No definite seepages have been recorded except in a small area on Bear River near the mouth of Columbia River. In prospecting this region bore holes should be sunk upon the axes of the anticlines and the venture should be regarded as purely experimental.

On the northern flank of the Olympic Mountains an extensive series of lower Miocene formations of sedimentary origin occur. These deposits consist exclusively of shales and subordinate amounts of sandstone and conglomerate. They are entirely marine and in places are fossiliferous. They have been folded and to a certain extent eroded. In most places they are mantled with glacial drift. No direct indications of oil have been seen in this region although there is no evidence to prove conclusively that it is not present. Provided bore holes were sunk upon the axes of the anticlines drilling would be warranted from an experimental standpoint.

On the western border of the Olympic Peninsula there is present a sedimentary formation possessing a thickness of at least 10,000 ft. which is certainly pre-Tertiary in age. This assemblage of strata is termed the Hoh formation. It is impossible absolutely to determine its age. It may be Cretaceous, Jurassic, or even older. In the northern Cascades and on Vancouver Island the Triassic and Carboniferous formations are metamorphosed. The Hoh formation is altered to only a very minor degree. It is much more indurated than the Chico or upper Cretaceous as exposed on the northeastern flank of Vancouver Island. It resembles in lithologic appearance somewhat the Jurassic rocks on Vancouver Island. However, lithologic characteristics form a poor basis for correlation. This formation extends from Point of Arches, just south of Cape Flattery, southward to Queets River and from the Pacific Ocean easterly in to the center of the Olympic Range for an undetermined distance. The formation is composed predominately of dark-gray shales and sandy micaceous shales with subordinate amounts of gray medium-grained sandstones which in places are gritty. Occasionally bands of conglomerate are present. Some of the shale bands are often composed of alternating belts of thinly bedded sandstones and shales the individual members of which are only a few inches in thickness. The entire series of strata have been thrown into anticlinal and synclinal folds. Two of these folds trend nearly north and south parallel to the coast. On the flanks of these major folds minor undulatory folds or warps have been developed. All of these structures have been deeply dissected by erosion and upon the eroded surface extensive deposits of fluviatile sands and gravels have been laid down. Later these have been cut into by the present rivers flowing westerly to the coast. Along the banks beneath the overlying gravels the strata of the Hoh formation are exposed in places. Good exposures may also be seen in the cliffs along the ocean. No fossils have as yet been found in this formation with the exception of fragments of carbonized wood, and the remains of diatoms and foraminifera. No traces of molluscan remains are known to occur. At a number of localities small seepages of oil may be seen in the shaly sandstones. At many other local-

ities a pronounced odor of petroleum may be detected even at some distance from the rock exposure. About 2 miles north of the mouth of Hoh River in western Jefferson County and just south of Hoh Head a seepage occurs on the beach. One-fourth mile inland the odor is extremely noticeable. Several pits have been made there and a shaft has been sunk to a depth of 20 ft. In the bottom of the shaft a small amount of oil seeps out. This has been collected from time to time and several analyses have been made. The results show the oil to have a paraffin base and low specific gravity. About 7 miles to the east another seepage occurs. Along the coast northward to Quillayute River localities where the odor of oil is pronounced are numerous. At a number of places along Bogachiel River similar exposures may be seen. It was the presence of these indications which originally gave incentive to the search for petroleum in commercial quantities. In 1901 and 1902 an attempt was made to drill for oil south of the mouth of Quillayute River on the coast. The efforts were unsuccessful. Apparently no attention was given to the structural conditions, as the well is located upon the flank of a syncline. In 1912, a well was started by the Washington Oil Co. at a point 1 mile south of Forks Prairie in Clallam County. At the present time it is said to be at a depth of 2,100 ft. The well is being sunk upon the axis of an anticline trending nearly north and south. Considerable amounts of gas were encountered at several depths.

Early in the following year a well was started by the Jefferson Oil Co., near the oil seep previously mentioned as occurring near Hoh Head. This well is being sunk on a short anticlinal fold trending northeast to southwest. The rock encountered to the present depth is mostly shale with alternating beds of sandy shale.

Farther south, just north of the mouth of Queniult River a well is being drilled close to a gas mound. The rock formation at the surface is upper Miocene. A low monoclinal fold trends northeast from the coast. Above it on the surface of the ridge about 300 ft. above sea level a small mound about 200 ft. in diameter exists. On the apex there is a small crater-like opening filled with water. Out of this gas is almost continuously escaping. Presumably the source is from the Hoh formation, which lies beneath unconformably below the upper Miocene.

It is possible that the Hoh formation extends into the heart of the Olympic Range. Reports are current of the presence of similar oil indications in that region. Whether petroleum deposits occur in commercial quantities or not in the Hoh formation has not as yet been demonstrated. The indications are sufficient to warrant a very careful examination and a series of test holes, provided they are sunk upon the axes of the anticlinal folds. There is absolutely no other possible means of predicting whether oil will be found in commercial quantities.

The origin of such petroleum as is known to be present cannot be absolutely proved. At a depth of 1,800 ft. in the Forks well, samples of shale show the presence of numerous skeletons of foraminifera. It is quite possible that these forms of organic life may have in part been the source. The shales near the Hoh River outcrops contain remains of diatoms. Some specimens of this shale upon microscopic examination are found to be crowded with diatoms. The soft organic portions of living diatoms are known to have fats as the organic constituents rather than cellulose. In the California districts diatomaceous shales are believed in many cases to have been the source of the oils. The shales bearing diatoms in the Hoh River region are not siliceous as in California. Diatoms compose only a small proportion of the rock. It is quite possible that the diatoms may also in part have contributed as a source for the oil. If oil deposits are ultimately found to be present in commercial quantities they may be expected to exist in the more porous sandstone zones. Such zones are present within the formation, as well as thick shale belts which can act as impervious cappings. Provided oil deposits are present, they would not necessarily exist beneath the areas where seepages occur upon the surface. Fractures in the strata might conduct such emanations of oil or gas to the surface over the axis of a syncline, which would be a very unfavorable place to drill. If oil is present in commercial quantities there is no possible means at present of predicting at what depths oil-bearing zones may be expected to occur. The same zone is certain to vary in depth at different localities.

Conclusions

From the geological conditions known to exist in the Cascade Mountains and in eastern Washington the chances for the occurrence of petroleum deposits in commercial quantities are very unfavorable. The Eocene formations in the Puget Sound Basin have been formed under conditions which would not allow organic material to accumulate as a source of petroleum. The overlying lower Miocene strata which were involved in a great anticlinal fold have been so extensively eroded that even though at one time oil deposits may have existed within them they would long since have been destroyed as the result of erosion. Conditions are thought to be unfavorable for the occurrence of petroleum deposits in the Puget Sound Basin.

In Thurston, western Lewis, Cowlitz, Pacific and southern Chehalis Counties, Eocene and Miocene formations are present which are in part of marine origin and contain considerable quantities of marine fossils. Both shales and sandstones exist which could act as retainers and cappings. They have been folded into shallow folds. No seepages or direct indications of the presence of petroleum are known to occur.

There are no facts which would prohibit its occurrence and none to suggest its presence. Several wells are being bored in this region.

The only definite indications of the presence of petroleum within the State occur on the western side of the Olympic Peninsula in the Hoh formation. This formation is possibly Jurassic in age. The formation is about 10,000 ft. thick and has been folded into anticlines and synclines. At several places seepages of oil and emanations of gas are known to occur. The source of such oil as is known to occur may be derived from the remains of diatoms and foraminifera. Three wells are being drilled, the deepest of which is at a depth of 2,100 ft. Oil in small amounts definitely occurs, but whether it is present in commercial quantities or not can only be determined by drilling in those localities where structural conditions would permit of its occurrence.

DISCUSSION

MILNOR ROBERT, Seattle, Wash.—A year ago at this time, 12 wells were being drilled for oil in western Washington. Some excitement attended the operations; many thousand acres of land were leased to oil companies and stock was widely advertised. The location of the wells may be described by referring to the conclusions in Dr. Weaver's paper, in which the State is divided into four areas.

In the area described first, which includes the Cascade Mountains and eastern Washington, where conditions are decidedly against the existence of commercial oil fields, no new drilling was reported. In the second area, the Puget Sound Basin, where conditions are unfavorable, but to a less degree, five wells were being drilled. The third region, covering certain parts of southwestern Washington, contained four drilling wells and several others projected or beginning operations, in spite of the absence of petroleum seepages. None of the wells mentioned is fully active today.

On the western side of the Olympic Peninsula, along the Pacific Ocean, are three wells which have been the scene of operations for the past two to three years. The region they are prospecting contains seepages of light oils and is the fourth area described in the paper. The difficulties of transporting both men and supplies to the well sites have delayed operations and brought the costs up to high figures. The drilling of these three wells is interesting on account of the presence of oil in nearby seepages and its occasional appearances in the material penetrated.

A. C. McLAUGHLIN, San Francisco, Cal.—The most important part of Washington is on the Olympic Peninsula. The geological conditions on the Peninsula are not similar to the conditions in California, and it would not prove to be a continuous belt of oil-bearing territory from there to here if oil should be found at either end.

W. N. BEST, New York, N. Y.—What is the possibility of oil around Centralia?

MILNOR ROBERTS.—That is in the third district, where the Eocene and Miocene formations are present. The author regards that as neutral ground, without seepages, and with no indications of oil.

DAVID T. DAY, Washington, D. C.—Is the geologic structure there consistent with considerable accumulations of oil; and, secondly, have any bituminous shales been found in the Coast region of the State of Washington?

MILNOR ROBERTS.—I know of no bituminous shales in the whole region on the western side of the Olympics. The structure is considerably folded and somewhat faulted. It has variations in hardness and fineness of materials which might permit of cappings being present that would hold oil.

It would be impossible to say, in a general way, at what depth oil would be found, or at what depth capping would occur. The folds have put the cappings in such attitude that they can be studied very well, and borings would go to varying depths, depending on the local topography. No ideal sites have been found for drilling, but anticlines are numerous. The covering of glacial material makes prospecting arduous and rather unsatisfactory, since it hides the bedding.

J. B. TYRRELL, Toronto, Ont. (communication to the Secretary*).—Professor Weaver has discussed the possibility of the discovery of oil in different parts of the State of Washington. The occurrence of vast pools of oil on the Pacific slope in the State of California has stirred the imagination of many of those living farther north, and during recent years some exploratory work has been done in the province of British Columbia in the hope of finding some northern continuation or representatives of the pools which have yielded so much oil to our fortunate neighbors on the south. In prosecuting exploration in British Columbia, some investigations have been conducted throughout the country near the mouth of the Fraser River, and some interesting results have been brought to light by these explorations.

In the southwestern corner of British Columbia, through which the Fraser River flows westward to the sea, there is a comparatively flat-lying area bounded on the north by granite mountains of Jurassic age, and on the east by the Sumas and Chilliwak Mountains, composed of somewhat older rocks. To the south this area extends across the International Boundary Line into the State of Washington, a rather high ridge of glacial débris extending east and west near the International Boundary Line. This comparatively small triangular area of about 700

* Received Sept. 29, 1915.

square miles in Canada is underlain by Tertiary sandstones and shales with a total thickness of several thousand feet. These sandstones and shales do not appear to contain any fossil remains of marine animals, but they do contain trunks and fragments of trees and land plants which have been altered to lignite or low-grade coal, this coal occurring at numerous horizons throughout the formation.

The mode of formation of these Tertiary beds would appear to have been about as follows: In pre-Tertiary times a great submerged valley extended along to the south of the Jurassic Mountains. Into the head of this submerged valley or bay the early representatives of the Fraser River and other streams emptied, carrying down into it loads of detritus worn from the upper portions of the river valleys, and this detritus was spread over the floor of the bay, while around the margin of the bay grew forests whose trees fell into the edge of the water. While the streams were thus discharging their loads of sediment, the land was gradually sinking and the sediment was gradually accumulating to a greater and greater thickness. The trees growing on the shore fell into the water and were also included in this sediment, and finally formed irregular beds of lignite of greater or less extent, each bed marking the position of an old shore line of the time when its vegetation was buried. Thus the bay was gradually filled up with beds of sand and gravel, with included seams of coal or lignite, having a total thickness of several thousand feet.

After these Tertiary beds were deposited there was a rise of the land of 1,000 ft. or more, and the Fraser River cut out a canyon along the foot of the northern mountains to a depth of about 1,000 ft., and at the same time the surface of the adjoining land to the south was cut into hills and valleys, but there was no perceptible warping of the land in this district, and the beds retained the horizontal attitude in which they originally had been deposited. Subsequent to the formation of the canyon of the lower Fraser River, the land was again depressed to its present altitude, and the old canyon of the Fraser River was filled by glacial débris brought down from the mountains to the north of it.

At some time subsequent to the deposition of the Tertiary sands and clays, which are known as the Puget group, dikes of quartz-andesite were intruded through them near their northern border. These dikes also cut the beds of lignite, and subjected them to a process of distillation, giving rise to a dark oil which is still occasionally found exuding from the sandstone in the vicinity of the city of Vancouver and in the country to the east of it. It is not likely that this oil has any commercial significance or value, but its mode of occurrence as a distillate of lignite through the influence of hot dikes, is of interest.

The Occurrences of Petroleum in Eastern Mexico as Contrasted with Those in Texas and Louisiana

BY E. T. DUMBLE,* B. S., HOUSTON, TEX.

(San Francisco Meeting, September, 1915)

THE history of the several petroleum deposits of Texas and Mexico, or of the sediments in which they now occur, if the deposits are not indigenous to such sediments, is known in a general way only, but it nevertheless casts strong light on the conditions surrounding them and is of great importance in the search for and exploitation of them.

The petroleum deposits of Texas are separable into two main divisions:

1. The deposits of northwestern Texas, which are directly connected with the Oklahoma fields to the north, and have no representatives at all in the Mexican area.

2. The deposits of the coastal area, which extend into Louisiana on the east and connect with those of the Mexican coast to the south.

Early in the Pennsylvanian period a series of bituminous shales and limestones were laid down over a broad area in Texas, Arkansas, and Oklahoma. These are known as the Bend series in Texas, the Caney shales in Oklahoma, and the Fayetteville beds in Arkansas. Following their deposition an uplift occurred, forming the Lampasas geanticlinal, which runs northeastward from the old Paleozoic land area of the Llano region toward Red River.

The Bend series is petroliferous in places and has furnished a few producing wells on each side of the Lampasas uplift. This uplift forms the barrier and dividing line between the two divisions of the Texas petroleum deposits.

To the west of it, in waters which were the southern extension of the interior continental seas, in connection with the sediments and coal beds of the Pennsylvanian and Permian, great quantities of petroleum were deposited either as oil or as "potential petroleum." These sediments have been subject to very slight disturbance, and only very gentle folds are found, with no signs of volcanic action in any part of the area. The conditions, therefore, indicate that the greater part, if not all, of the oil in this area is indigenous to the beds in which it is found.

East of the anticlinal the oldest beds which appear in contact with the Bend are the Trinity Sands of the Lower Cretaceous. No evidence has

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been found either on the surface or in wells of the presence of sediments referable either to the Coal Measures, Permian, Trias, or Jura.

To the southward, in Central Mexico, very complete sections are found of both Trias and Jura, but if the waters of those periods ever reached the Texas coast, no evidence now remains to prove it.

During the Cretaceous, however, the southern seas did cover a part of the Texas area, and the sediments of its earlier period (the Lower Cretaceous or Comanche) in the two areas are closely related. They consist of sands, clays, and limestones, which in the coastal area present little, if any, evidence of being oil bearing.

The beginning of the Upper Cretaceous deposition marked a very decided change in this respect and ushered in a period in which the formation and deposition of petroleum or petrolic matter was widespread through the coastal area.

Böse,¹ describing the section between San Luis Potosi and Tampico, mentions the limestones of the Tamasopa Gorge and their exposures in Micos Canyon, saying of them that they are probably Cenomanian or Vraconian. They form the top of the Middle Cretaceous deposits of the Mexican classification and are overlain by shales and marls of their Upper Cretaceous.

By far the greater portion of all the oil so far found in the Tampico-Tuxpam area is contained in or derived from this Tamasopa limestone.

During the period in which this Tamasopa limestone was being deposited in the Mexican region, farther north in east Texas and Louisiana shallower waters prevailed and in them the sands and clays of the Timber Creek beds or Woodbine sands and clays were laid down. The Cenomanian age of these beds is indicated by Stanton,² who states that this fauna "contains species of *Acanthoceras* and other types resembling those that are characteristic of the Cenomanian in Europe."

Böse³ also correlates these Lower Cross Timber sands with the uppermost portion of his Middle Cretaceous.

These Woodbine sands are the principal source of the extensive oil production of the Caddo and other fields of western Louisiana.

Between this great Louisiana field on the north and the greater Mexican field on the south there is an interval of more than 600 miles in which these formations are not now found within the coastal area, unless some portion of the basal Eagle Ford shale may represent a time equivalent, and even if that be the case, no oil deposits are found.

Immediately overlying the Tamasopa limestone there is a series of

¹ *Guide Book, International Geological Congress*, 1906.

² A Comparative Study of the Lower Cretaceous Formations and Faunas of the United States, *Journal of Geology*, vol. vi, No. 6, p. 606 (Sept.-Oct., 1907).

³ *Neue Beiträge zur Kenntniss der Mex. Kreide*, p. 10 (1910).

limestones and clays known as the San Juan or San Felipe series and these are overlain by the Papagallos or Mendez shales.

The San Juan limestones have yielded a few imperfect ammonites, which may belong to the Taylor marl or the Austin chalk.

No fossils have ever been found in the Papagallos shales, which constitute the latest Cretaceous deposition in the Mexican coastal area and a facies of it not represented in Texas.

This divergence in geologic history began during the period of the Austin chalk with the appearance of a barrier south of Eagle Pass in the vicinity of the Sabinas River. This barrier was extended southeastward to the Gulf coast near Tordo Bay, so that in place of a continuous strand line, as at present, there were for a time two separate basins, the southern one being directly connected with the waters of the Pacific Ocean. This condition continued until the end of the Eocene; and the shales of that age found overlying the Papagallos shales south of the Tamaulipas barrier carry fossils of the California types, rather than of the Texas types such as occur in the region north of it. These shales were followed by other shales, yellow clays, sands, and limestones which carry marine invertebrate fossils throughout, those of the lower portion being of Oligocene age, while those of the upper beds are Miocene (?). It is evident from the similarity of the deposits that the conditions of deposition varied very little in this area after the close of the Eocene, when the waters of the Gulf succeeded those of the Pacific.

During this Upper Cretaceous and Tertiary time the conditions existing along the Gulf coast north of the Tamaulipas barrier were entirely different. Marine deposits alternate with those of palustrine and flood-plain origin, and with periods of dry lands and æolian deposits, showing here continued variation instead of the uniformity of the Mexican area.

The history of oil deposition in the two areas is as different as that of their geology. With the close of the Cenomanian, the formation and deposition of petroleum seems to have greatly diminished in the Mexican region south of the Tamaulipas barrier, and few, if any, workable deposits of indigenous oil have yet been found in the beds of the Upper Cretaceous, or of the overlying Tertiary.

North of the Soto la Marina, since the beds are in large part direct continuations of those in Texas, there may be a similarity in such oil deposits as exist; but drilling in this area has not yet proceeded far enough to permit any definite statement to be made regarding them.

In Texas and Louisiana, however, the processes of oil formation were active at various times and pools of oil or gas are found at several horizons of the Cretaceous between the Woodbine sands and the Eocene, as well as in different members of the Tertiary. While the oil in some of them may not be indigenous, in many of them it seems to be so.

Eastern Mexico

Tuxpam-Tampico Field.—The Tamasopa limestone as exposed in Micos Canyon is composed of massive beds of hard limestone with abundant fossils. In drilling wells nearer the coast we find that while the beds of this series are still massive, they include strata in which the lime is more or less arenaceous and beds of sand, suggesting a coast line to the eastward during the Middle Cretaceous. The oscillations and land conditions of the beds of the Coastal Plain here included in the Middle Cretaceous are further shown by the occurrence in them of beds of gypsum sand of considerable thickness.

The San Juan series, which is largely thin bedded limestones in the type locality, is represented in this area by shales and shelly limestone beds with occasional heavier beds of limestone as well as of sand. They are characterized by the brown color of part of the shales and limestones and have a thickness of 400 to 800 ft.

The Papagallos shales are usually blue or gray in color, but occasionally have red bands, and the variable thicknesses shown in different wells seem to indicate a considerable erosion of these beds prior to the deposition of the overlying Eocene shales. The total thickness in this area of the Papagallos is estimated at 700 to 1,200 ft.

The Alazan shales are similar in character to the Papagallos, and it is difficult to distinguish them apart, either on the surface or in the wells, unless fossils are found in them; and, so far as we can judge by drilling, the weathered yellow surface clays in the upper members of the section in the central portion of the area are derived from similar gray marls, shales, and shells of lime. In the southeastern portion of the territory, however, somewhat different conditions of deposition seem to have existed, as was shown by a well drilled near Tuxpam to the depth of 3,787 ft. In this well the upper 500 ft. was composed of sandy materials corresponding to the Tuxpam beds, but the balance of the well was shales and gumbos. Nothing was found which could be identified with the San Juan.

The rocks of the coastal area have a general easterly dip of low angle. Near the western border, along the foot of the Cordilleras, they are strongly folded and faulted, but these folds and faults gradually diminish toward the coast. Intrusive rocks, chiefly of a basaltic nature, occur with these sedimentaries as laccoliths and sills, and in many places have reached the surface and now appear as hills and necks.

The most valuable oil pools are found along the anticlinals or domes or in faulted areas, of which the anticlinals may be the continuations. While strong seepages may occur in the vicinity of some of the volcanic necks, representing the residuum of oil which has reached the surface through fissures caused by the eruptive forces, the best wells have usually

been found away from such pronounced disturbances. The basalt occurring in the producing oil fields is mostly in the form of sills or intercalated sheets of moderate thickness.

The principal deposit of oil is that of the Tamasopa limestone. The producing pools found in the San Juan series and overlying shales are largely, if not entirely, migratory oil from the Tamasopa, which has risen to its present location through faults or fissures, and the strongest flows are apparently secured along such line of fault or fissure. There are small deposits of indigenous oil at various horizons in these shales, both of the Cretaceous and the Tertiary, but there is nothing that would indicate the occurrence in them of such oil in workable quantities.

In the Tuxpam-Tampico field we have, therefore, a vast body of indigenous oil in the Tamasopa limestone, overlain by dense beds of marls and shales which have been folded and faulted by orogenic action, permitting the collection and concentration of the oil in domes and anticlinals and along the fissures and fractured areas of the limestones and, through these, in the porous beds of the overlying shales. The intrusive and extrusive material accompanying these orogenic movements is basalt. While the movements apparently began during the Upper Eocene, the greater portion of the basalt extrusives are seemingly of late Tertiary or Quaternary age.

North of the Tamaulipas barrier there are two probable oil fields: The San Jose de las Rusias, which is a continuation of the Tampico field, and the Northern or Soldadito, which is a continuation of the Texas Coastal field.

San Jose de las Rusias.—The surface of the region lying east of the Tamaulipas Range and south of the Soto la Marina River is occupied principally by Oligocene deposits known as the San Fernando beds, through which the eruptives of the San Jose de las Rusias Range have forced their way. In this area there are seepages of oil in such quantity as to warrant careful drilling; and since beds of the Eocene seem to be wanting and the San Fernando beds occur in direct contact with the Papagallos shales, the Tamasopa limestone should lie within reach of the drill.

Soldadito.—The coastal strip lying between the Rio Grande and the Soto la Marina is occupied by deposits which are direct continuations of those north of the Rio Grande; and similar indications of oil, sulphur, and gypsum are present at certain localities. The only well of which we have record, although it did not exceed 750 ft. in depth, showed both sulphur and oil.

Northwestern Texas

In this field we have an example of the occurrence of indigenous oils which is of interest from the fact that the beds containing them, compris-

ing the entire series of deposits of the Carboniferous and Permian periods in this area, show comparatively slight effects of earth movements and absolutely no traces of vulcanism. The petroleum deposits, therefore, having been subjected only to minor extraneous causes of change, are as nearly as possible in the condition of their original deposition or formation. Low domes and low broad anticlinal or monoclinal folds furnish the collecting places for the commercially valuable pools; and the surface indication of these is often so slight as to require instrumental surveys to determine their presence. Oil pools are found in sands of the Wichita, Cisco, Canyon, Strawn, and Bend series. The character and occurrence of the oil sands are well described by Udden:

"The position of the sands in the entire section suggests that they are ancient sand bars and perhaps beach sands, built up, washed away, rebuilt, and finally buried under accumulating argillaceous sediments, during a long period of more or less gently changing geographic conditions, involving, on the whole, a progressive sinking of the shore-land and the adjacent bottom of the sea. Sands connected in one place may in another place be separated. Closely contiguous sands may be wholly separate. Sands clearly interrupted at some point may be connected by some devious circuit in an unknown direction."⁴

This field brings out strikingly a relationship between coal beds and oil deposits, which, if it exists elsewhere, is not so apparent.

The Cisco division of Cummins carries Coal Seams 7 and 8 of this section. These are found outcropping from Bowie eastward through Jack and Young Counties. Near the Colorado River in Coleman County, these seams have been opened up and are known there as the Chaffin and Bull Creek seams; and the exact position of these coal beds as regards fossiliferous horizons of this series is well shown by Drake's report⁵ on the Colorado Coal Field of Texas.

A well drilled west of Henrietta on the Halsell farm to a depth of 3,985 ft. furnished a series of samples and fossils which were studied by Udden. The results⁶ show that the well penetrated the entire Coal Measures series of the Pennsylvanian and entered the underlying Bend. The relationship of the oil and coal horizons is thus described:

"It appears probable that the horizon of the Chaffin coal in the Cisco division corresponds to the dark blue shale reported as underlying the gray lime at from 1644 to 1655 feet below the surface in the Halsell boring, and that the Bull Creek coal is the stratigraphic equivalent of what is reported here as dark blue slate lying from 35 to 70 feet below the 'gray lime' at from 1420 to 1436 feet below the surface. It will be recalled that Drake reports having observed *Fusulina cylindrica* about 150 feet above the Bull creek coal in considerable abundance,⁷ and that this is the uppermost part in

⁴ Bulletin No. 23, University of Texas, p. 100.

⁵ Fourth Annual Report, Geological Survey of Texas, pp. 357 to 446 (1892).

⁶ Udden: Bulletin No. 23, University of Texas, pp. 82 to 83.

⁷ Report on the Colorado Coal Field of Texas, Drake and Thompson, Fourth Annual Report, Geological Survey of Texas, p. 413 (1892).

the Colorado section from which he reports this fossil. Likewise the limestone at 1445 feet below the surface in the Halsell well is found to contain this fossil, and no rock higher up in this well seems to be of a kind in which this fossil is at all likely to occur, excepting the other thin limestone reported at the depth from 1420 to 1436 feet. This part of the Halsell well section is doubtless also the equivalent of the deeper productive oil and gas sands in the two fields under investigation. These consist of shales, limestones and sandstones, which lie at from 1500 to 1700 feet below the surface in the wells near Petrolia and at from 1800 to 2000 feet below the surface in the Electra field. This general correlation seems to be warranted by palaeontologic evidence as well as by evidence based on the lithologic character of the beds explored by drilling."

This well is probably 30 miles or more from the outcrop of the coal; and we cannot now say just what part of that distance is underlain by the extension of the coal bed before its replacement with the oil shales and sands, or how near the coal and oil deposits are to each other.

Coal seam No. 1, in the Strawn division of Cummins' section, has been worked for many years near Thurber and Strawn; and recent drilling has found oil a few miles west of the mines. The well records here, as studied by Kennedy, show that this coal maintains its thickness for some 3 or 4 miles west of the present workings, and that it then gradually thins to the west, is finally missing entirely, and oil is found in the same horizon in sands and shales. The distance between known coal and known oil is only about 500 ft. Furthermore, a well $2\frac{1}{2}$ miles farther west shows bituminous shales occupying this horizon; and neither coal nor oil was found in them.

The conditions here seem best explained by Murray Stuart's idea of the sedimentary deposition of oil. There was seemingly a coal swamp or lagoon along the sea coast, the petrolic decomposition of the organic débris of which furnished the oil which was carried out and deposited in the contemporaneous sediments of the sea floor.

The Coastal Area

Cretaceous.—While some impregnations of asphalt occur in Burnett County in connection with certain beds of the Trinity sands, these are probably remnants of migratory oil from the underlying Bend series. Similar migratory oil occurs in the same sands in Montague County. No other occurrences are reported from beds of the Lower Cretaceous.

In the typical section of the Upper Cretaceous of central Texas the Timber Creek or Woodbine beds were followed by the Eagle Ford, Austin, Taylor, and Webberville beds. In eastern Texas and Louisiana the lithologic character of these beds is somewhat altered, and other names are used to designate their representatives.

The active deposition of Cretaceous oil began with the Woodbine series. These sands outcrop from Woodbine south of Red River to the Brazos near Waco; but in this region, if they were ever petroliferous, the

oil has been forced out by the inflow of water, and now they are water-bearing beds, and no signs of petrolic deposits are known in them.

South of the Brazos the Woodbine is overlain by the Eagle Ford and Austin and does not appear at the surface. Shallow wells on the South Bosque River west of Waco, drilled in 1890 by Col. William L. Prather, found small quantities of light oil in the Woodbine. This field was developed in a small way later, and is the only occurrence from these sands so far recorded in central Texas.

From Woodbine the outcrop stretches eastward into Louisiana, and in the Caddo region the sands are highly petroliferous and furnish the best wells of that belt. The productive oil and gas of the Eagle Ford is confined to the "Blossom sand," which is unknown as such on the Colorado. Gas is plentiful in the Blossom sand in the Caddo field, but such oil as is found there is heavy, and producing wells are rare if not unknown. The bituminous and even petroliferous character of the shale is apparent, however, at many places along its outcrop in central Texas, and it furnishes seepages of a heavy tarry oil from small wells near Waters Park, north of Austin.

The Annona chalk, which is an eastern representative of a portion of the upper Austin or basal Taylor, is a good oil horizon, and the Nacatoch sand, probably the equivalent of part of the Taylor, is an excellent gas horizon. Thus, in Louisiana, there are four distinct horizons of gas and oil in the Upper Cretaceous, the principal of which is the Woodbine, and it is probable that the oil and gas in these beds are largely indigenous.

The deposits of commercial value occur in connection with folds and domes, and at present are best developed in the area between the Sabine and Red Rivers. Harris's idea is that the great accumulation of oil in this area is due to an extensive crustal movement, which he calls the Sabine uplift, and which has brought the Upper Cretaceous beds to within 700 ft. of the surface over a large area.⁸

West of San Antonio a few wells drilled in Medina County developed small quantities of oil in the Austin chalk; and it is probable that later these may prove to be of value.

Near Cline, west of Uvalde, there is a deposit of asphaltum which occurs in the Anacacho limestone. The Anacacho of this area may be the equivalent of the Annona of the East Texas section.

The other occurrences west of the Sabine River are confined to the Taylor beds. The pools at Powell, Corsicana, San Antonio, Thrall and other localities find their supply in these beds. In some of these the oil is possibly indigenous, but at Thrall it is probably migratory.

Toward the end of the Austin there was an outbreak of vulcanism in this area and active volcanoes were in operation around Austin. Pilot

⁸ Oil and Gas in Louisiana, *Bulletin No. 429, U. S. Geological Survey*, p. 27 (1910.)

Knob and other necks are evidence of their character and the mingling of the volcanic ash with the sediments of upper Austin and lower Taylor gives the date of their activity.

From drilling so far done at Thrall it would appear that just after the close of the Austin there was at this place a flow of vesicular lava (probably on the floor of the sea) which was later covered by the sediments of the Taylor. It is in connection with this eruptive material that the Thrall oil occurs. The fact that this eruptive action took place, and the similarity of the oil to that of the Woodbine, suggest the possibility that the oil may have originated in the Woodbine sands and reached its present location through rifts and fissures made by volcanic action in the firm sediments of the Eagle Ford and Austin.

At present we have no direct evidence bearing on the duration of activity of these volcanoes, but there is nothing in the surroundings of the old necks, so far as known, to suggest that they were centers of eruption throughout any extended period. The Pilot Knob volcano, however, marks the beginning of a series of earth movements that continued at intervals well into the Pleistocene. The effects of the earliest known succeeding movement are found in the numerous Cretaceous domes and ridges of eastern Texas and Louisiana, which are, seemingly, of pre-Tertiary age, since the domes include beds of Webberville age and are surrounded unconformably by the sediments of the lower Eocene.

A later movement caused the Sabine uplift, which involves not only Cretaceous beds but lower Eocene as well.

In connection with the Jackson and Oligocene of eastern Texas, we find several extensive deposits of volcanic ash, which shows the existence of volcanoes or submarine fissure eruptions near enough to permit such accumulation; but it can hardly be supposed that these were derived from the known volcanoes of the Cretaceous.

Eocene.—The deposits of Eocene age, in spite of their wide areal distribution in the Gulf Coastal Plain of Texas and Louisiana, have yielded comparatively small amounts of petroleum.

No oil whatever is known in the Midway or Wilcox of the Lower Eocene and only two small pools have been found in the Marine beds which overlie them. At Oil Center, Nacogdoches County, a small body of oil of a lubricating grade was found, and another small amount of similar oil was found in these beds at Crowther in McMullen County, some 300 miles southwest of the first. While oil indications may occur at other points, no workable deposits are known.

The Yegua sub-stage of the Claiborne, however, has proved to be a very valuable gas horizon in the region between the Sabine and the Rio Grande; and we confidently expect that it will equal in productiveness some of the sands of the Cretaceous, or even those of the Carboniferous, when it shall have been properly exploited.

Beds of the Jackson and Oligocene, which followed these, are equally destitute of oil.

Harris found a few Jackson fossils in two wells at Sour Lake and a few others were found in a well at Saratoga. The relations of these beds to those of the surrounding Neocene, however, show that their presence is due to folding or faulting; and it is probable that any oil they may carry is migratory, since the well at Saratoga penetrated the Jackson sediments for a hundred feet and found no indications of oil in them, but instead, a sand carrying a very large volume of sulphurous water of high temperature. The only oil found in this well came from a series of blue shales, thin beds of marly limestones and sands, about 200 ft. above the top of the limestone carrying Jackson fossils. It may be remarked here that although a number of wells were drilled in the same vicinity to depths over 200 ft. greater than the depth at which the Jackson limestone was found, these wells all ended in the same blue shaly clays and marly limestones without any appearance of this heavy fossiliferous limestone. At Sour Lake the same conditions exist. Jackson fossils were found in a well at the depth of about 1,500 ft., but no oil was found in it. Wells drilled to much greater depths than 1,500 ft. within the territory lying between Saratoga and Sour Lake do not show beds of the Jackson stage.

The probabilities are, therefore, that while the conditions were favorable for the formation and deposition of vast quantities of lignites and an abundance of other organic matter during the Eocene, the conditions for the formation or storage of petroleum were comparatively unfavorable.

Neocene.—The oil of the Gulf Coast proper all occurs in sediments of Neocene age, often in or around domes, and frequently in connection with deposits of salt, gypsum, and sulphur.

The Neocene section was first differentiated on the Nueces River in southwest Texas, where the basal sands and clays carrying, in their higher beds, fossils of Loup Fork or Upper Miocene age were called the Oakville. The sands overlying these, with Pliocene fossils of Blanco age, were called the Lapara, and succeeding these were the Lagarto clays and the Reynosa. Neither the structure nor the character of these Oakville or Lapara beds lends itself to the formation or retention of petroleum. In the south central extension of the country underlain by them several wells have been drilled completely through these deposits without showing any signs of either gas or oil. Taking them as a whole, they are non-bituminous and may be looked upon as dry.

East of the Brazos this exact lithologic succession has not been found, and in east Texas the term Fleming clays has been used for a series of deposits which, as exposed on the surface, are of palustrine and fluvial origin, and, as evidenced by the fossils collected from them, may represent the entire Neocene section below the Lafayette or Reynosa.

The shore deposits of the Fleming, as shown in surface exposures, are

for the most part blue and greenish clays carrying great quantities of calcareous nodules, calcareous cemented sandstones, thin lenticular nodules of brownish to brownish yellow ferruginous sandstones, and occasional heavy deposits of the same material, and of arenaceous limestones. The vertebrate remains are mostly found in the blocks or large boulders, but sometimes appear in the clays.

The seaward extensions of these deposits are very different from the shore deposits. Well records show below the sands, gravels and clays of the Lafayette:

1. A series of brown shales or shaly clays with streaks of gumbo. These have a thickness of 300 to 500 ft. and possibly represent the Woodville.

2. Thin to heavy bedded interstratified shales, sands, and gumbo, with shells of rock and some gypsum, underlain by massive beds of gumbo, blue and brown shale, gravel, and sand, probably representing the Burkeville, 1,000 to 1,500 ft.

3. Interbedded shales, sands, limestones, and gumbos similar in character to those of the upper portion of the preceding series, but occasionally of more massive bedding. These probably represent the Navasota-Oakville Miocene horizon, and have a thickness of more than 1,000 ft. Some portions of these beds, as shown by fossils found in them, are of marine origin, but the exact relations of the marine and palustrine deposits have not been ascertained.

These beds thicken considerably toward the coast and complete sections are only found in undisturbed areas. In the vicinity of domes or folds some portions, if not all, of these beds are usually lacking, such as may have been deposited having been removed by erosion.

These beds are fossiliferous, but unfortunately the pieces of shell coming up in the drilling are usually very small and unidentifiable, although at some points fairly large pieces of small oysters are obtained. At Saratoga, however, shells are found in the shales at 1,200 ft. which are identical with those of Galveston Deep-Well Miocene at 2,400 ft.; and similar shells are also found in wells at Batson.

The shore deposits of the Fleming carry a fair-sized fauna. In the region around Alexandria in Louisiana the fauna is altogether of brackish-water forms, but at Burkeville we have a mingling of brackish-water forms with vertebrates; and at Navasota, where the deposits are in a great measure calcareous clays and limestones, they carry vertebrate remains and rolled Cretaceous shells. Some of the deeper-water shales and clays carry the Burkeville brackish-water fauna, and it is found at Pine Prairie at a depth of 1,545 ft., at Anse la Butte at 1,550 ft., at Edgerly at 3,000 ft., and at Terry at 3,100 to 3,300 ft.

Many of the invertebrates found at Burkeville and eastward are new, and there is little that is useful for exact determination of age. Dr.

Dall considered them Pliocene.⁹ No other determinable invertebrates were found in the series.

The vertebrates were studied by Dr. Matthew, who concluded that they embraced species from Middle and Upper Miocene and possibly Lower Pliocene.¹⁰ The older beds occurring in the Trinity and Brazos drainage carry vertebrates of Upper and Middle Miocene age, and, therefore, probably represent the Oakville of west Texas. Teeth of *Protohippus* or long-crowned *Merychippus* were found in the same beds with the Pliocene shells near the base of the section at Burkeville; and it is possible that this portion of the beds is Pliocene and corresponds in age to the Lapara of west Texas, while the higher horizon at Woodville may represent the Lagarto, which it closely resembles lithologically.

While the shore deposits of the Fleming beds are non-bituminous and no sign of either oil or gas has ever been found in them, the seaward extensions are bituminous to a considerable degree. Small pieces of lignite and asphaltic material have been reported from a widely extended series of wells. Shows of oil appear in these beds at various horizons and good wells have been obtained in them. Wells in the Saratoga field drilled to a depth of over 1,200 ft. obtained their oil from these shales, and the wells at Terry get their oil from the Burkeville horizon. The large producers along the eastern side of Humble and the western side of Sour Lake fields appear to obtain their supplies from sandy shales belonging to this series.

The Lafayette, which closes the Pliocene in this area, shows in some wells a thickness of 500 ft. of sands, gumbos, and clays. They carry water in abundance, but no oil.

Of the overlying Pleistocene deposits the Beaumont clay, of Port Hudson age, is most characteristic. These clays and sands are very variable in thickness. In places the Pliocene beds are found in low hills, surrounded but not covered by the clays, while in others the Beaumont clays show a thickness of 2,500 ft. and more, above these beds.

The probabilities are, as stated, that all the oil of the Louisiana and Texas coastal belt is of Neocene age. It is separable into two classes, shale oil and dome oil. The first is regarded as indigenous to the beds in which it is found, and the second as migratory oil derived from it.

The relations of these two classes of oil are particularly well shown in the Humble field.

The dome oil was first discovered, and the field brought in, in 1904. Development and comparative exhaustion were very rapid, the climax of production, 90,000 bbl. per day, being reached in June, 1905, after which it declined to 20,000 bbl. by December of the same year. The

⁹ *Water Supply Paper No. 335, U. S. Geological Survey, p. 73 (1914).*

¹⁰ Dr. W. S. Matthew, in letter.

underground conditions were the same as those of other dome fields; the wells were comparatively shallow, being usually less than 1,400 ft., and the oil, like that of the other dome fields, was of 20 to 24 Bé gravity. The plug of underlying salt was found at 1,400 to 1,600 ft. over an extensive area.

As the production of this central pool declined, wells were sunk at various distances from it; and the Esperson wells, a mile south of the dome, found a pool of light oil in shales. Later, similar oil was found in shales north of the old field, and more recently large producers have been secured in the shales from 1 to 2 miles east. This oil has a gravity of over 30° Bé.

As shown by the well records, these shales are found on the west, north, and eastern sides lying well up on the gypsum and other mound-forming material, but not surmounting the same.

On the western side, a well about $1\frac{1}{2}$ miles northwest of the main portion of the field or dome was drilled to a depth of 3,015 ft., of which the last 1,000 ft. were gumbos, rock, and shales, with some sand and streaks of gypsum. These beds evidently belong to the series of shales, gumbos, and rocks we are now considering. At the depth of 2,200 ft., a small show of oil was found, and small quantities of gas were met with at various depths down to 2,800 ft.

On the south side of the San Jacinto River and north of the original field, a number of wells have been drilled to depths varying from 2,700 to 2,900 ft. These wells have been small and irregular producers, obtaining their supplies from what has been termed a broken sandy shale, or thin sands interstratified with thinly bedded or laminated shales.

Along the eastern and southeastern sides of the Humble field wells have been drilled to depths exceeding 3,300 ft. In a well on the Stephenson Survey the gumbo, rock, and shale beds appear to begin at about 2,400 ft. These are the producing beds from which all the outlying wells on the north, northeast, east, and southeastern sides of the Humble field obtain their supplies.

How close to the central portion of the Humble dome these shales approach is not known, but in the Crosbie well on Block 12 of the Landslide the same shales occur at a depth of 1,500 ft. and in the Esperson wells on the Hermann tract in the Jones League at a depth of 1,650 ft.

These shales appear to thin out toward the dome. Thus, in the deep wells on the southeastern and eastern sides they appear to have a thickness of at least 1,000 ft., while in the Esperson and Crosbie wells nearer the dome they have thinned to 700 ft., and disappear altogether in the Landslide or Cherry tracts before reaching the Echols ridge or mound proper. Along the north and western sides the same thinning and disappearance are found.

Similar conditions exist in the Sour Lake field, at Spindletop, and elsewhere.

From the condition in which these shales lie and their relation to the mounds it is clear that the salt and gypsum and other beds forming the mound-making materials are much younger than the shales. Not only does the general tilting show this, but it may be observed that at many places the salt itself has been found intruding into the shales. This is the case at Anse la Butte, Vinton, South Dayton, and Hoskins Mound.

The origin and manner of formation of the domes of east Texas with their great bodies of salt, gypsum, and sulphur have been the subject of much discussion, but without an entirely satisfactory explanation being reached.¹¹

Whatever the causes back of them may be, they became operative in late Cretaceous time, were active in the Eocene, and again in the Neocene and Quaternary, as is proven by wells drilled in and around the domes of the different belts. From the evidence it appears quite certain that the Sun mounds near Waller and Damon's Mound are post-Lafayette.

The very variable thickness of the Port Hudson, as already noted, evidences the extremely irregular surface of the Pliocene deposits on which the Beaumont clays were laid down, a condition which is in great contrast with the present level surface of this area.

Possibly nothing could bring out the differential movements that have taken place in this coastal area since the Miocene deposition more clearly than the fact that while the Pliocene brackish-water fauna which occurs at Burkeville 150 ft. above sea level is found in abundance at Terry, 66 miles south, at a depth of from 3,000 to 4,000 ft., the marine Miocene fauna which occurs at Saratoga at a depth of 1,000 ft. is only 2,400 ft. deep at Galveston, 74 miles south of it.

It seems clear, therefore, that during the Neocene the coastal area of east Texas was subject to extensive oscillation, and it is probable that these movements rather than those of earlier date were directly connected with the formation of the domes and folds found here.

The relation of the bodies of salt, gypsum, and sulphur of these domes to the surrounding sediments indicates that these masses have certainly penetrated 2,000 or 3,000 ft. of the sedimentary strata. The clays, sands, and limestones immediately adjacent to or overlying them are tilted at comparatively high angles for this region, the surrounding sedimentaries dip away from them at lower angles, and beds or sills of salt and gypsum extend from the main mass out into the surrounding beds. The great similarity existing between these occurrences with their bosses of salt

¹¹ A late paper on the subject which seems a step in advance of former ones is by E. G. Norton, *Origin of the Louisiana and East Texas Salines*, *Trans.*, li., p. 502, (1915).

and gypsum and those of the intrusive basalts of the Mexican field is well worthy of attention.¹²

The origin of the oil found in the coastal area has been variously explained. Its possible derivation from beds of the Permian or Cretaceous age has been suggested under the hypothesis that the oil reached its present horizon through faults formed in connection with the uplift of the domes. These fissures must have extended through the overlying Eocene and Neocene sediments for this to have been true.

We have no evidence whatever of any Permian deposits southeast of the Lampasas geanticline nor of the continuation of the Woodbine as an oil horizon so far southward as the coast.

The lowest horizon reached in any of the oil-pool wells is the Jackson, which shows no signs of oil. Below this there is, above the Woodbine, at least 2,000 ft. of Cretaceous and 2,000 ft. of Eocene strata. These are mostly uncompacted sediments with many water-bearing beds under a heavy head. Any fissure line which would make a passage for oil from the Woodbine to the upper sands would be equally open to water from these various intermediate beds, and, judging from the results of similar conditions in well drilling, the water would soon drown the oil and close the fissure by caving.

If such transmission of oil were possible we should find some evidence of it in the Caddo region (where such fractures also occur) in the presence of oil in various porous beds above the Woodbine, but this is far from true. Such oil as is found above the Woodbine gives no suggestion of having migrated upward from these beds.

Another strong presumption against the pre-Neocene derivation of the oil is that we know of no oil in the Eocene, and the oil of the Cretaceous has a paraffin base, while all of the coastal oil has an asphalt base.

The close association of the oil, gypsum, salt, and sulphur in some of the domes has naturally suggested the idea of a common origin or a close relationship in origin, and this has been widely discussed. Were this true, the oil should be found in connection with all such domes, and such is not the case. We have oil pools where there is no dome and where no salt has been found, and we have numerous domes and bodies of salt, gypsum, and sulphur without any accumulations of oil.

A simpler solution by far, and one which seems fully justified by the facts, is that the oil of this area was formed in the usual manner, deposited in beds of Neocene age, and was collected into pools and concentrated by the movements which may also have given rise to the domes. When the uplift which caused one of the domes was in the vicinity of one of these pools it captured a part or all of its oil, and where there was no pool near it the dome had no oil.

¹² Garfias: The Effect of Igneous Intrusions on the Accumulation of Oil in Northeastern Mexico, *Journal of Geology*, vol. xx, No. 7, p. 666 (Oct.-Nov., 1912).

Conclusion

Possibly the greatest contrast between the occurrences of oil in Mexico and those in Texas and Louisiana is, that while the Mexican deposits so far as developed practically originated in a single horizon, the indigenous oils of Texas and Louisiana not only occur in great quantity at a similar horizon, but are found at many other horizons, both below and above, ranging from the Pennsylvanian to the Pliocene.

The Mexican oil occurs in a field which has been subjected to the action of strong orogenic forces, the intrusive and extrusive material associated with the movements being basalt.

The Texas and Louisiana fields embrace deposits in areas in which evidence of orogenic action is very slight, and in which no intrusive or eruptive materials are known, as in northwest Texas; those in which it has been greater, and in which the eruptive or intrusive materials are basalt, as in central Texas, and rock salt, as in western Louisiana; and finally of those which have been subjected to much greater movements, associated with which, as intrusive and protrusive materials, we find salt and gypsum. With these sulphur is connected, possibly as a secondary product.

The Mexican oil pools are apparently of greater extent and the conditions surrounding the occurrence of oil in them are far more uniform than those of the coastal oil fields of Texas and Louisiana. Consequently, the questions arising in connection with the discovery and exploitation of the oil are much simpler and easier of solution.

Taken as a whole, the Mexican field will probably show a greater aggregate area of oil pools, will furnish wells of greater volume and will be longer lived than the fields of Texas and Louisiana.

DISCUSSION

E. L. DEGOLYER, Norman, Okla. (communication to the Secretary*).—Referring to Mr. Dumble's description of the stratigraphy of eastern Mexico, I should like to question the classification of the Papagillos or Mendez shales as being of Cretaceous age and the so-called overlapping of the Papagillos or Mendez shales by the Alazan shales; the classification of the San Felipe series as Cretaceous, and the correlation of the San Felipe series with the San Juan limestones.

Mr. Dumble apparently recognizes the continuity of the great shale formation by his use of the terms "Papagillos" and "Mendez," as equivalent and interchangeable, since the Papagillos Hills in the State of Nuevo Leon and Mendez Station in Northern Vera Cruz, localities at which the shale formation outcrops, and from which the formation names are taken, are approximately 250 miles apart and on opposite sides

* Received Oct. 18, 1915.

of the Tamaulipas barrier. The Papagallos shales (evidently referring here to the northern extension only) are said to be non-fossiliferous and no reason for their Cretaceous classification is given, though in a previous article¹³ they are classified as Cretaceous because at their northernmost outcrop they "have the same relation toward the overlying Tertiary that the Escondido (Cretaceous) has through its whole extent."

The Alazan fossil locality was found by me and the first collections from it were sent to Dr. Dall; about 17 species of marine invertebrates were recognized by him, the fauna noted as having a distinct Pacific tincture, and the age of the formation from which they came pronounced lower Eocene. The outcrop of the shale formation has been traced continuously from this point by me and my assistants into the Panuco River region where Eocene fossils are reported at a depth of 1,810 ft. in a well at Topila.¹⁴ From Topila the formation has been traced continuously north to the Padilla-Jiminez region in Central Tamaulipas. Lithologically, the formation is fairly constant and though I am only generally acquainted with it north of the Padilla-Jiminez region, until definite evidence of the Cretaceous age of the Papagallos shale in its type localities, or of an overlap, is available, it seems to me more logical to consider the shale formation a single stratigraphic unit of Eocene age.

Such fossil evidence as is known to me regarding the age of the San Felipe series is meager and not very exact. Fossils from a depth of 3,150 ft. in Tamijuin well No. 3 were examined by Messrs. Hopkins and Belt and were said to have a decided Tertiary aspect. These fossils came from the San Felipe series as nearly as can be determined from the well log. Fossils from a depth of 2,900 ft. in Ganahl well No. 1, at the junction of the Moctezuma and Tamuin Rivers, were pronounced to be of Tertiary age by Dr. Hart. There is little doubt but that these fossils were from the San Felipe. On these classifications, the stratigraphic position of the formation, and lack of contrary evidence, I should classify the San Felipe as being of lower Eocene age. In addition to this evidence there are certain lithologic and stratigraphic features which have a bearing on such classification. The San Felipe series lies between the Cretaceous limestones and the known Eocene shales in northern Vera Cruz. It resembles a transitional series, being composed of alternating impure limestones and shales, often sandy, the limestones predominating near the base of the formation and the shales increasing toward the top until the formation grades so gradually into the overlying Eocene shales that it is almost impossible to determine the exact contact between the two formations where they outcrop and, in my opinion, quite impossible to determine the contact accurately from a well log; but it is apparently

¹³ *Science*, New Series, vol. xxxiii, No. 841, pp. 232 to 234 (Feb. 10, 1911).

¹⁴ E. T. Dumble: *Science*, New Series, vol. xxv, No. 910, pp. 906 to 908 (July 7, 1912).

separated from the underlying Cretaceous limestone by an unconformity. Basal conglomerates, containing pebbles of Cretaceous limestone, in the San Felipe have been reported from several points in southern Tamaulipas and northern Vera Cruz and there is some evidence of the removal of the upper part of the Cretaceous limestones before the deposition of the San Felipe series in certain localities.

The San Felipe series apparently overlaps the San Juan limestones in southern Tamaulipas and its equivalent in north-central Vera Cruz, the San Juan extending perhaps into the Panuco-Topila region. The San Juan is absent from the excellent Boca del Abra section between Valles and Las Palmas, San Luis Potosi, and in the Potrero del Llano—Juan Cassiana region; the San Felipe series rests directly upon the massive bedded Tamasopo limestone.

The Furbero Oil Field, Mexico

BY E. DEGOLYER*, A. B., NORMAN, OKLA.

(San Francisco Meeting, September, 1915)

I. INTRODUCTION

THE Furbero oil field lies in the Gulf coastal plain of Mexico between the important ports of Tampico and Vera Cruz and about one-third of the distance between the two points from the latter (Lat. $20^{\circ} 22' N.$, $97^{\circ} 30' W.$ Gren.). It is in the State of Vera Cruz, about 14 miles west by south of the village of Papantla in the canton of the same name, and is the most southern of the developed oil fields of the Tampico-Tuxpam region being a distance of 50 miles south by east from the Potrero del Llano field and 42 miles in the same direction from the Alamo field. The field has been developed by the Oil Fields of Mexico Co. and is in the midst of a large block of lands owned or leased by them. Communication is through the port of Tuxpam with which it is connected by a narrow-gauge railway and 6-in. pipe line some 52 miles in length.

This field is of more than ordinary interest since it represents a type of oil deposit hitherto unknown. So far as I am aware, no other commercially valuable deposit of oil is known to occur under conditions of accumulation similar to those found here. The igneous nature of the so-called 'sand' encountered in drilling was not recognized until a number of wells had been drilled; and it was not until recently that the geological relations of the sill or laccolith, which gives rise to the accumulation of oil, were known and the so-called 'anticline' was known to be, in part at least, the result of igneous intrusion.

Geological surveys in this general region were made in 1900-1901 by W. H. Dalton and in 1907-1909 by P. C. A. Stewart, F. W. Moon, Leonard V. Dalton, and Dr. Max Muhlberg. More recent surveys have been made, however, under the direction of the author, by E. L. Ickes, B. C. Belt, B. Hartley, and Harve Loomis. Notes of the earlier examinations were not available with the exception of the report by Dr. Muhlberg who apparently first recognized the true nature of the Furbero intrusion. This paper is indebted to his work and that of the more recent surveys for much of the detail regarding stratigraphy and to Messrs. Percy Furber and Arthur C. Payne, President and General Manager of the Oil Fields of Mexico Co. for permission to publish, and for much of the in-

*Chief Geologist, Cia. Mexicana El Aguila, S. A., Cuban Oil Co.

formation regarding the field. The detailed structure of the intrusion as herein set forth is the result of a study of the well logs and samples by the author.

II. HISTORY AND DEVELOPMENT

This general region, including Furbero, is one of the earliest known oil regions of Mexico. Indian tradition says that this part of the coast was settled during the seventh or eighth century; the Sabaneta topographic basin, in which Furbero lies, being then known as Choloa. That the existence of oil seepages was probably known to the early inhabitants,

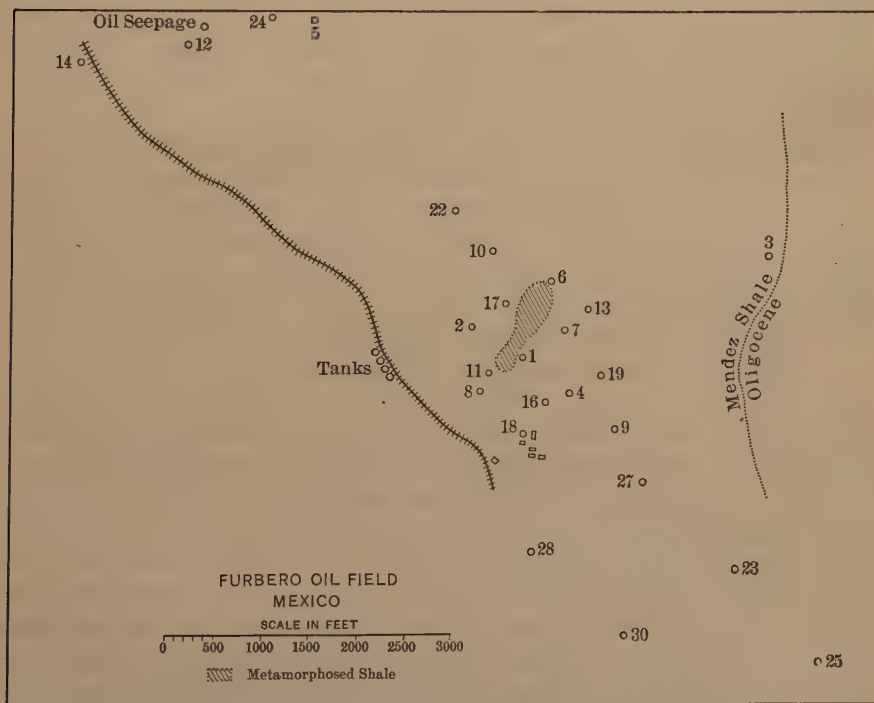


FIG. 1.—MAP OF FURBERO OIL FIELD, SHOWING DEVELOPMENT.

is indicated by the fact that the locality now known as Furbero was formerly called "Cougas" (later corrupted to "Cubas"), a Totonaco word, meaning a species of black wax or oil. Moreover, Spanish writers immediately following the conquest note "chapapote"¹ as among the

¹ "Chapapote" is the term by which asphalt, petroleum residue, and even oil itself are known in Mexico at the present day. It is derived from the Aztec words "tzauc," meaning paste or cement and "popochili," meaning perfume and probably refers to two of the early uses of the material. Robelo, Cecilio G.: *Diccionario de Seudoaztequismas*, p. 536 (Cuernavaca, 1906).

articles for sale in the great market of Mexico City. Old workings consisting of a shaft walled up with rock were found near one of the principal seepages by the earliest known white explorer.

In 1868, a certain Dr. Autrey, who had been carrying on the practice of his profession at Papantla for several years, discovered the larger seepages, and the knowledge of their existence and location being made public, a company (the Cia. Explotadora de Petroleo del Golfo Mexicano) was formed at Mexico City in 1869, for the purpose of exploiting the deposits. Machinery was brought into the region and a 3-in. bore-hole was sunk to a depth of 125 ft. near one of the largest seepages. No production resulting, a tunnel was driven 50 ft. into the hill near the old Indian workings, and a moderate supply of oil (reported as about 4 or 5 bbl. per day) was secured. The project was soon abandoned, however, and about 1870 the properties, including abandoned machinery, were denounced by Autrey. One of the old stills was set up in Papantla and the crude oil was packed mule-back to that point where some 4,000 gal. of kerosene were refined and sold locally.²

In 1883, the present mining law, vesting the ownership of petroleum rights in the ownership of the surface of the land, was enacted, and the property thus passed out of Autrey's possession. In 1892, El Cougas and 2,500 hectares of land were leased to an American oilman, this being the first oil lease of record in Mexico.³

From 1895 to 1900, numerous leases were made in this general region, and in 1901 the Mexican Petroleum and Liquid Fuel Co., Ltd. was formed and took them over. This company was founded with English capital, among the interested persons being Cecil Rhodes of South African fame. During the years 1901 and 1902, drilling was actively carried on, 24 wells being drilled, two of which attained a depth greater than 1,500 ft. Operations being unsuccessful from a commercial standpoint, the region was abandoned. In 1903, the Oil Fields of Mexico Co., the present owner of the Furbero field, was organized. Lands and leases over a great area were secured. Machinery was brought into the region via the Tecolutla River and wagon roads; and in 1904 drilling was commenced in earnest. In 1907, the first well "came in"; and in April, 1910, the pipe line and railroad to the port of Tuxpam having been completed, commercial deliveries of oil began. Since that time drilling and oil production have been almost continuous. Thirty-two wells have been drilled by this company in the general region, 24 of them in the Furbero field proper.

² This early history is taken from a copy of a letter dated Mar. 2, 1892, from Adolpho Autrey, M. D., to John F. Dowling, City of Mexico.

³ Cuestiones Petroleras de Actualidad, Mexico, 1912; Lic. Edward Schuster, p. 1.

III. TOPOGRAPHY

The region in which the Furbero field occurs is topographically a great basin shut in on the north, west, and south by the great lava-capped mesas which lie at the foot of the central Mexican plateau (the remnants of a flow which once covered the entire region), and on the north and east by Tertiary hills. The major axis of this basin runs northwest and southeast and the drainage is controlled by two master streams: (1) The Cazonos River, which emptying into the Gulf of Mexico about 40 miles northwest of Furbero, drains the northern part of the basin, including the Furbero field proper, through its tributary the Arroyo Coachepea and secondary streams; and (2) the Tecoluta River, which empties into the Gulf at a point 30 miles east of Furbero, draining the southern part of the basin. The water-sheds of these streams are separated by a very low divide. The Furbero field proper, as developed hitherto, has an area of little more than 100 acres. It lies in the northeastern part of the basin on the slope from the Tertiary hills to Arroyo Coachepea on the west. The most striking features of the immediate topography are the sharp twin hills in the midst of the field, which resemble very much the small volcanic plugs or necks so commonly features of the topography of the oil regions north of the Tuxpam River. The general elevation is from 600 to 650 ft. above sea level.

IV. GEOLOGY

This region is covered, for the most part, with dense tropical jungle and a mantle of soil resulting from centuries of vegetable decay; and as a consequence there are but few rock exposures of geological value. These are commonly found in the banks of creeks or arroyos. No exposures large enough to admit the measurement of detailed sections of the various formations are found. We are therefore forced to depend mainly upon the soil and topographic expression, verified by occasional exposures, in mapping the surface geology, and upon well records and general regional observations in studying the details of the various formations.

In considering the Furbero field proper, we are directly concerned with the Tertiary marine sediments and the igneous rocks only; but a description of the Cretaceous limestones is here given for the purpose of defining the relation of the Furbero field to other fields of the region.

1. *Cretaceous*

Tamasopo Limestone.—The oldest known rocks of this general region consist of a series of limestones, known from their outcrop in the Tamasopo Canyon, State of San Luis Potosi, as the Tamasopo limestone. This

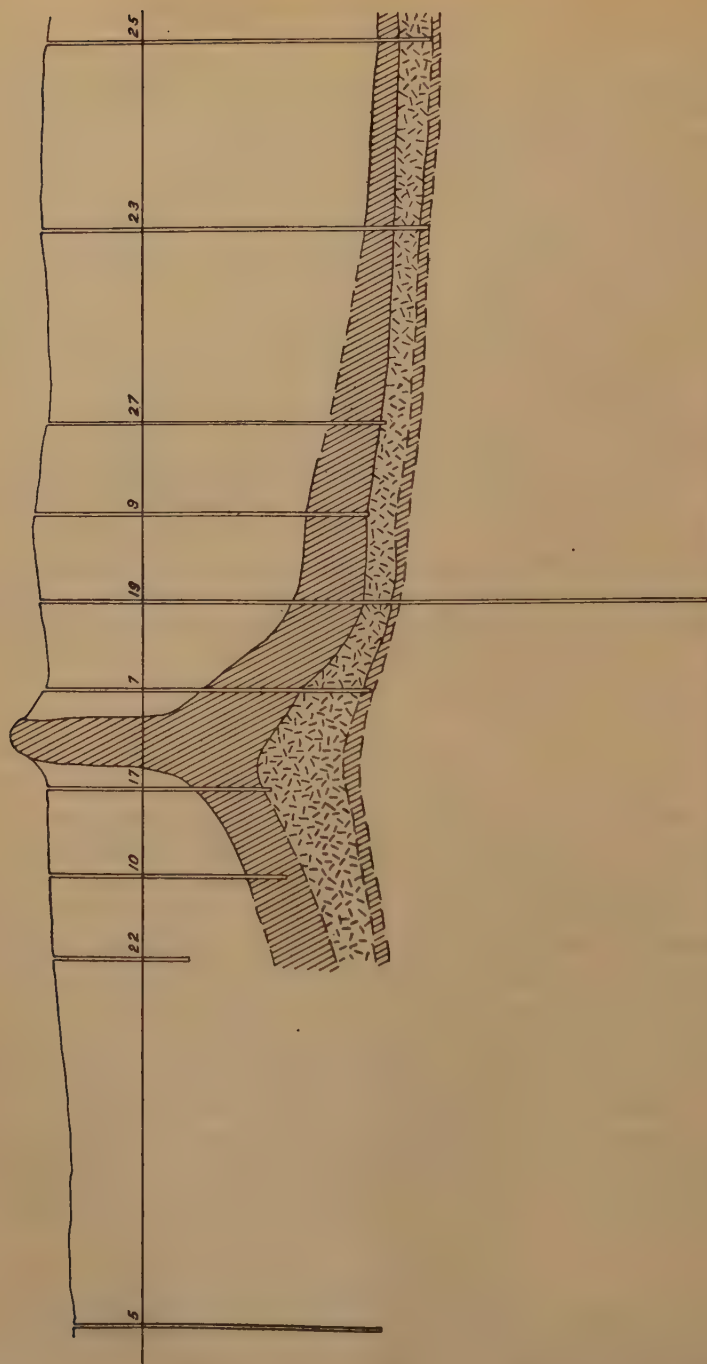


FIG. 2.—GEOLOGICAL SECTION OF THE FURBERO OIL FIELD.

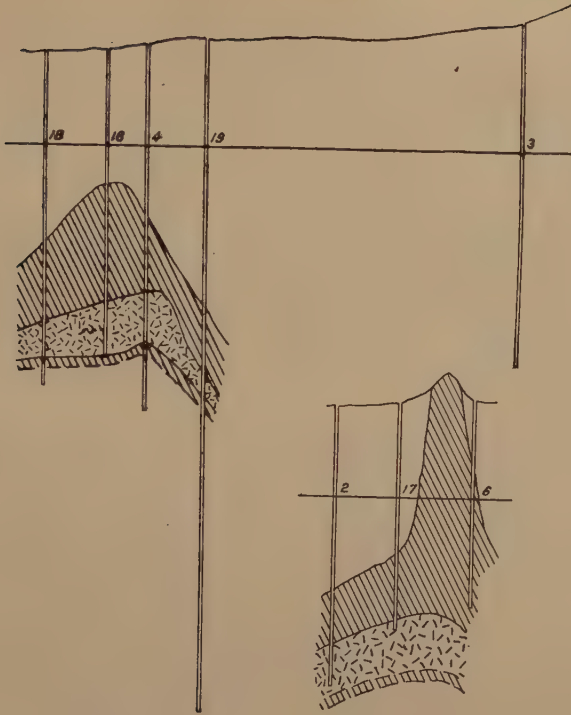


FIG. 3.

FIG. 4.

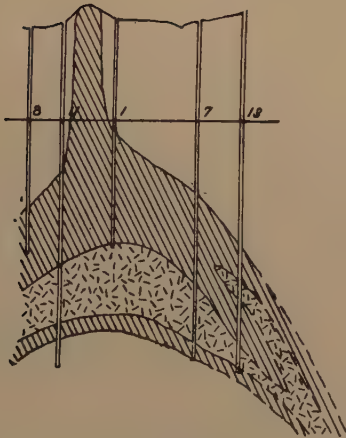


FIG. 5.

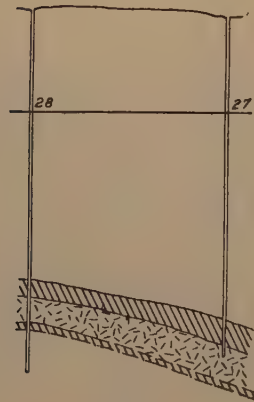


FIG. 6.

FIGS. 3 TO 6.—GEOLOGICAL SECTIONS OF THE FURBERO OIL FIELD.

formation consists of hard, gray, pure, compact, porcelain-like limestones bedded in layers less than a foot thick, and is characterized, in its upper part, by the occurrence of an abundance of black to dark gray and green chert nodules, interbedded with the limestones. Economically, the formation is of greatest importance because of its oil-bearing horizons. The uppermost members of the limestone, which are massively bedded in the northern Vera Cruz and Valles regions, are somewhat porous and contain great solution caverns. These cavernous limestones are the reservoir rocks of highly productive fields of Potrero del Llano, Juan Casiano, Dos Bocas, Los Naranjos, and Alamo, which are characterized by immense and prolific gushers (daily capacity 50,000 to 100,000 bbl.). The formation is generally regarded as the parent oil-bearing formation of northern Mexico, from which the oil at present found in other formations has come by migration.

The main mass of the outcrop of the formation in this region is in the Sierra Madres, the front range of which passes points respectively 28 miles west and 16 miles south of the Furbero field. The formation has not yet been encountered by wells drilled in this region, but it is very probable that the oil of this field originated in the Tamasopo and was forced upward by hydraulic and gas pressures, along a channel formed by the broken and metamorphosed zone of contact of igneous and sedimentary rocks, or through faults and fissures, to the reservoir from which it is now obtained.

The thickness of the formation in this region is unknown, but probably is from 6,000 to 8,000 ft. The age of the formation is lower Cretaceous, evidently the equivalent of the Comanche of Texas.

2. Tertiary Eocene

San Felipe Beds.—Overlying the Tamasopo limestone and resting unconformably(?) upon it, is a series of alternating, impure, thin-bedded limestones and gray, red, and green shales and marls known as the San Felipe beds. The entire formation is somewhat sandy and contains, locally, beds of tuff of variegated colors which contain decomposed mica and are finely porous. It is also reported to contain conglomerate though well records do not show such a rock. The economic importance of the formation in this region has not yet been determined. It has been encountered by two of the wells drilled and oil and gas have been produced from it, but not in great quantities. In the fields of the Panuco River valley near Tampico, it contains the oil-producing horizons. With the exception of one or two doubtful inliers, the outcrop of the formation in this region is confined to a narrow strip adjoining the outcrop of the Tamasopo limestone in the mountain front. The thickness of the formation varies from 600 to 1,000 ft.; it is probably more nearly the latter in this region. The exact age of the formation is unknown but it is apparently

Tertiary if one may judge from the few fossils which have been secured from drill cuttings. If such is true, it is of lower Eocene age. The formation grades imperceptibly into the overlying shales series, the limestones gradually becoming more argillaceous and impure and grading finally into hard shale and, in turn, into soft shale. It is doubtful if the upper 100 to 200 ft. of the formation can be differentiated from the overlying shales in well logs.

Mendez Shale.—Grading from the underlying San Felipe beds, is a thick series of gray to green shales, marls, and clays, containing rarely thin shaly sandstones and limestones and red shales, called the Mendez shale, because of its outcrop near Mendez station, just west of Tampico. Economically, it is generally of no importance; but in the Furbero field, part of its ordinarily impervious shales have been baked and metamorphosed into a hard, brown to black, porous shale and it forms, together with the crystalline igneous rock, the oil-reservoir rock of the field. Apparently much of the metamorphism has been due to the action of ascending thermal waters associated with the igneous rock after its intrusion; for the pipe of metamorphosed shales which rises to the surface and outcrops in the twin hills in the central part of the field cannot have been the direct result of the heat of the igneous rock. The igneous rock of the intrusion has also undergone alteration since crystallization.

This formation outcrops for the most part over the entire floor of the Sabaneta basin. It is the surface rock of the Furbero field proper, extending from the Oligocene hills on the east to the lava flows at the foot of the hills of the Sierra Madre on the west. The thickness of this formation at Furbero is approximately 4,000 ft. No fossils have been found in this region; but a collection made at Alazan some 50 miles to the north has been identified by Dr. Dall as lower Eocene.

3. Tertiary: Oligocene

Overlying the Mendez shales is a thick series of sandstones, shales, impure fossiliferous limestones and occasional conglomerates of Oligocene age. The various strata making up the formation are lenticular and grade laterally into each other. Near the front of the Sierra Madre occur beds of shale so thick that their outcrops are hardly distinguishable from those of the Mendez shales. The formation is not known to be of economic importance in this region. The local outcrop occurs over an area east of a N-S line just east of well No. 3.

4. Igneous Rocks

The igneous rocks of the region consist of basalts, dolerites, basalt-gabbros, and various products of volcanic activity, such as volcanic sands,

ash, etc. The surface outcrops consist almost altogether of lava-capped mesas, the remnants left by the erosion of a great lava flow which once covered most of the region. Several big dikes are found near Reparo and many porphyritic basalt dikes cut the Oligocene rocks near Buena Vista. A patch of volcanic ash covers the surface near Palma Sola. The lava flows evidently originated at some point in the Sierra Madre, probably in the vicinity of Necaxa. The flow varies in thickness from a few feet to 500 ft., and it has a uniform inclination of several degrees to the northeast, an inclination due evidently to the topography of the pre-existing land surface. The nearest remnants of this flow to the Furbero field are the lava-capped mesas of El Brinco, 6 miles to the northwest, Mecatepec, 9 miles north, and a mesa near Tecuantepec River, 12 miles east of south. There are no evidences of deformation since these flows, which occurred after the deposition and folding of the entire series of marine sediments.

Of greater interest and economic importance is the occurrence of a laccolith or extremely thick sill, the formation of which, together with associated phenomena, has given rise to the accumulation of oil known as the Furbero pool. This intrusion does not reach the surface. The only indication of its existence is the outcrop of metamorphosed shale already mentioned. It has been well explored, some 16 wells having been drilled into it, five other wells into the overlying metamorphosed shales, and six wells, encountering no evidence of intrusion, in the immediate vicinity. The intrusion evidently commenced in the form of a sill following closely the bedding planes, the sedimentary rocks having already been folded into an anticline. The intrusion thickened near the crest of the anticline, lifting the overlying beds and further accentuating the already existing fold. Small amounts of oil are produced from the igneous rock in several of the wells. As has been observed, the gabbro seems to have been greatly altered and is quite porous in certain localities. The greatest thickness of igneous rock shown by the drill is in well No. 7 where 440 ft. were drilled through. A careful construction of the probable form of the intrusion, however, suggests a maximum thickness of from 600 to 650 ft. near wells 17 and 7. Minimum thicknesses of 193 ft. and 170 ft. have been encountered in wells 19 and 28 respectively, though the actual thickness of the intrusion is probably little more than 100 ft. at well 19; the greater apparent thickness being due to the fact that the intrusion is dipping steeply where encountered by the bore, which therefore does not show a true section. The highest point reached by the intrusion, as shown by well records, is in well No. 17 where the top of the intrusion was reached at a depth of 1,490 ft. or 891 ft. below sea level. The lowest point yet reached is in well 25 at a depth of 2,665 ft. or 1,984 ft. below sea level. The general form is indicated by the accompanying sections. Two sills of igneous rock separated by some 300 ft. of meta-

morphosed shale were passed through by well No. 13. The area known to be underlain by the igneous rock as determined by wells which have been drilled up to the present time (encountering either the igneous rock itself or the metamorphosed shale) is 26 acres; but the actual area underlain is known to be greater than this, probably several times as large.

5. *Structure*

The general structure of this region, as of the Gulf coastal plain for some distance northward, is that of an eastward-dipping monocline; the oldest rocks—the Cretaceous limestones—outcropping in the steeply folded and faulted ranges of the Sierra Madre and passing under the Tertiary rocks of the coastal plain, and the Tertiaries outcropping generally in the order of their succession, the youngest beds fringing the coast. The beds of the monocline are gently folded, the complexity and steepness of the folding and faulting being greatest in the Sierra Madre, and decreasing with distance from it. Folding took place in Oligocene or post-Oligocene time and afterward the entire region was subject to igneous activity. As has been noted, it was almost covered by an immense lava flow, the remnants of which cover hundreds of square miles at the present time. The sedimentary rocks were also intruded to some degree; but the surface evidences of intrusion are not very extensive.

The general structural strike of this entire region is northwest-southeast, the axes of the minor folds of the Tertiary rocks of the coastal plain being parallel to those of the steeply folded Cretaceous rocks of the Sierra Madre. One of these minor folds, the Furbero anticline, having been complicated by the intrusion of igneous rocks and the metamorphism of previously impervious beds, affords the structure in which the oil of the Furbero pool has accumulated. It may be held by some that the folding of the sedimentary rocks has been due to the intrusion. That they may have occurred contemporaneously is possible; but that the intrusion represents an incident of the more general igneous activity which took place in the Tampico-Tuxpam region after the folding of the sedimentary rocks, seems more probable. That the intrusion followed generally pre-existing structure seems probable. The major axis of the Furbero anticline and the major axis of the intrusion are the same and parallel to the axis of the other folds in the general region. The floor of the intrusion is of a shape generally anticlinal and is evidently influenced in form by conditions existing at the time of the intrusion. Had the strata been level when the igneous rock was intruded, one would have expected to find the laccolith with a flat floor, the arching of the beds being confined to those above the plane of intrusion. The dip of the upper surface of the laccolith, though in the same direction as that of the floor, is generally

steeper; but the exact effect of the intrusion on the position of the outcropping strata cannot be determined because of scarcity of surface exposures. It seems probable that the intrusion ascended from below in the form of a dike, and encountering a bedding plane of little resistance which was dipping steeply (west flank, sections *B* and *C*) was diverted into that plane, and took the form of a sill. The splitting of the intrusion (well 13) probably indicates the point of this change of form.

V. DETAILS OF DEPOSIT

As will be noted from the accompanying map, the laccolith has been intensively explored, several of the wells being only 80 to 100 ft. apart and most of them being within 200 ft. of other wells, so that several have been found to interfere with each other in actual exploitation. Wells more recently drilled in the exploration of the southern part of the laccolith have been located from 800 to 1,000 ft. apart.

This field is structurally a somewhat complicated anticline, and the occurrence of oil may be classed as conforming to the structural theory; yet the determining factor in the accumulation of oil has been the providing of pore space for a reservoir by the metamorphism, accompanying the igneous intrusion, of the under- and overlying shale, which were previously impervious. Of consequence also has been the subsequent alteration of both shales and igneous rock; but, as this was probably an indirect result of the igneous activity, it need not call for separate consideration. Normally, the unaltered sediments were probably quite similar to the unaltered shales into which they grade, both the altered and unaltered shales being of the Mendez formation, a series of blue and gray, medium-soft, fine-grained shales, more or less calcareous in places, and (when not metamorphosed) of fairly constant lithological character throughout. The metamorphosed shales seem to be of two general types: A fairly hard, black shale, differing but little from the unaltered gray shales into which it grades; and a hard, brown, porous, silicified shale, which usually occurs immediately above the igneous rock and grades into the black shales. The zone of metamorphosed shales envelopes the igneous rock which it both overlies and underlies, and is distinctly thicker over the intrusion than under it. Above the apex of the intrusion the metamorphosed zone continues in the form of a pipe or chimney to the surface, where its outcrop forms two sharp hills, resembling in contour the volcanic plugs of the region north of the Tuxpam River. This extension of the metamorphosed zone is probably due to ascending gases or thermal waters, and not directly to the heat of the intruded masses. As is seen on the sections, the upper metamorphosed zone including both the black and the brown shales, varies from 100 to 500 ft., and the underlying zone from 50 to 150 ft., in thickness.

The igneous rock is a dolerite or gabbro which has undergone, in places, a somewhat extensive alteration. Drill cuttings of the rock consist for the most part of gray sand much resembling a true quartz sand. This sand, together with the position and structure of the igneous rock, which somewhat resembles that of a thick lens of folded sedimentary rock, was quite conducive to concealment of the true igneous nature of the intrusion; and the geology was first interpreted as that of a simple fold in sedimentary rocks. Pieces of igneous rock several centimeters in diameter have been recovered from the cuttings. A sample from the intrusion penetrated by well No. 13 at 2,400 ft., has a porosity of 6 per cent., as determined by experiment. A sample from 1,835 ft. in well No. 16 is apparently a coarsely crystalline dolerite and is quite porous. A characteristic log of a well in this field as partly determined from well samples, is as follows:

Furbero Well No. 11	Thickness, Feet	Depth, Feet
Yellow clay (surface).....	29	29
Blue and gray shales with oily seams, between 400 and 900 ft. (unaltered).....	1,221	1,250
Black and brown shales of variable hardness (metamorphosed) with good shows of oil at 1,650 and 1,655 ft.....	415	1,665
Gray sand (igneous crystalline; good show of oil at 1,650 ft. con- tact).....	363	2,028
Black shales (metamorphosed).....	122	2,150
Gray shales (unaltered).....	200	2,350
Small production of oil at the upper contact zone of igneous and sedimentary rocks.		

VI. OCCURRENCE OF THE OIL

The oil in this field occurs in commercial quantities both in the igneous and metamorphosed rocks. Well No. 27, the largest well in the field, though it was already producing while in the upper metamorphosed shales, "came in" with an initial production of almost 1,000 bbl. per day upon being drilled 30 ft. into the igneous rocks. It has produced more than 200,000 bbl. of oil, most of which has apparently come from the igneous-rock reservoir. Well No. 28 also showed an initial production of 150 bbl. per day from a depth of 150 ft. into the igneous rock. Well No. 9 has produced approximately 150,000 bbl. from the upper metamorphosed shales; and most of the production of the remaining wells has come from the metamorphosed shales, the zone of contact with the igneous rock being particularly productive. Salt water has been encountered immediately under the oil in several of the wells.

As has been observed, the greatly varying porosity of the rock, rather than the structure, is believed to have been the determining factor in the accumulation of the oil of this deposit. The most productive wells of the field have been Nos. 9 and 27, and they lie to the east of the center of the area of greatest production. The crest of the laccolith or dome lies altogether outside the most productive area, though some oil has been encountered in every well drilled to a depth at which it might reasonably be expected. In my opinion, the intrusion would have been as effective in any form which it might have assumed, provided that it did not outcrop.

From a knowledge of this deposit and of the Mexican fields in general, I believe that the oil of this deposit originated in the Cretaceous limestone, and has been forced upward to its present position along the metamorphosed zone of contact of the sedimentary and igneous rocks.

The Mexican Oil Fields

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I. HISTORY OF OIL DEVELOPMENT IN MEXICO

THE occurrence of oil or "tar" in Mexico was mentioned as early as the seventeenth century by Friar Sagahun, who gives the Indian name "chapopote," by which these asphalt seepages are still called. This asphalt was apparently used, as it has been used by primitive people in many parts of the world, in religious ceremonies and for medicinal purposes. Travellers also report that the ruins in Yucatan and the pyramids in southern Mexico show traces of the use of "chapopote" as a building cement.¹

DeGolyer says that the first attempt to exploit oil or gas in Mexico in a commercial way is shown by the records of the Memoria de Fomento of 1865, when permission was granted to a Senor Ildefonso Lopez to exploit the deposits of petroliferous substances in the San Jose de las Rusias area in the State of Tamaulipas. Other concessions follow, and probably refer to surface seepages which occur in that district. This was several years after the discovery of oil in Pennsylvania. In 1868, a well 125 ft. deep was drilled by a company organized in Mexico City, in what is now known as the Furbero district, and a little oil was refined there.

In 1873, residents of Tampico denounced seepages along the Tamesi River, and asphalt was mined near Tempoal in the Canton of Tantoyuca. No drilling was attempted. But between 1880 and 1883 several shallow wells were drilled for oil in Mexico, two of them being near the present Potrero de Llano field west of Tuxpam. The wells were drilled by a Boston company with Canadian rigs, and are said to have reached a depth of about 400 ft., while one is reported to have flowed. Several other abortive attempts were made to drill for oil between 1885 and the beginning of the present century, about which time the Mexican Petroleum & Liquid Fuel Co., Ltd., in which Cecil Rhodes was interested drilled 24 wells unsuccessfully, several of them as deep as 1,500 ft., in the State of Vera Cruz. Also Messrs. Doheny and Canfield of California pur-

* Johnson and Huntley, Consulting Oil Geologists.

¹ De Golyer. *Oil and Gas Journal*, April 16, 1914.

chased land in the State of San Luis Potosi, and in May, 1901, the first successful well was struck in the present field of Ebano, 50 miles west of Tampico. Various shallow wells were drilled about this time in the States of Vera Cruz, Tamaulipas, near Guadalupe, Chiapas, and Tabasco on the Isthmus of Tehauntepec; and several asphalt companies attempted to mine the surface deposits in the Cantons of Tuxpam and Ozuluama.

In 1904, drilling was commenced by the Oil Fields of Mexico Co. (Pearson interest) at Furbero, and its first well was brought in successfully. Pez No. 1, the first large well on the Ebano property, was also brought in during the same year, and has flowed continuously since. About 1907, drilling became more active, especially in the Cantons of Tuxpam and Ozuluama. It was started at San Diego by the Pennsylvania Oil Co., at Tumbadero by the Pearson interests, and at Juan Casiano and Tres Hermanos (La Pithaya) by the Huasteca Petroleum Co. A Mexican bank drilled four shallow wells near the Tampalache seepage, just north of the Panuco field. Operations were also commenced by Pittsburgh interests at Los Esteros, in the State of Tamaulipas.

In 1908, the Pearson refinery at Minatitlan on the Isthmus of Tehauntepec commenced operations. In May of the same year San Diego No. 2, the discovery well of the Dos Bocas field, came in with a production of 2,500 bbl. daily. On July 4, the famous San Diego No. 3 was brought in, caught fire immediately, and burned until extinguished by encroaching salt water. The Chijol field in the Ebano district and the Topila field were brought in during this year, which inaugurated important operations in the Mexican oil fields.

The year 1910 saw the discovery well of the Panuco field (East Coast Oil Co. No. 401) drilled, and the first wells in the Tanguijo and San Pedro districts were put down in that year by the Pearson interest, the Mexican Eagle Petroleum Co. The Huasteca Petroleum Co. also brought in its No. 6 and No. 7 Juan Casiano, one with 14,000 bbl. and the other with 28,000 bbl. daily production. About the time these came in, this company completed its 8-in. pipe line from that field to Tampico. The Mexican Eagle Co., having completed its pipe line from Furbero to Tuxpam, brought in, in December, 1910, the famous Potrero del Llano well, yielding about 100,000 bbl. daily. Drilling was by this time very active; and in 1911 the first exportations of crude oil were made to the United States from Tampico.

In July, 1912, the Government put a tax of 20 centavos per metric ton (1.1 c. U. S. gold per barrel) upon all oil produced in the Republic. The construction of a large fleet of tank steamers had already been commenced by the Huasteca Petroleum Co. In 1913, the Penn-Mex Fuel Co., now a subsidiary of the Standard Oil Co. of New Jersey, through the South Penn Oil Co., brought in two good wells in the Alamo field near

Potrero del Llano. The Chila-Salinas and the Topila field began to suffer severely from the invasion of salt water. The Mexican Eagle Co. brought in the discovery well in the Los Naranjos field in Amatlan, which has since broken its valve and proved to be a much larger producer than was estimated by flowing at the rate of 50,000 bbl. per day for a week. On December 1, the Government raised the production-tax to 75 centavos per ton (approximately 4c. U.S. gold per barrel) besides increasing numerous duties affecting the petroleum industry. A bill was proposed in the Chamber of Deputies for the nationalization of the petroleum industry, but no decisive action was taken.

During January, 1914, the Dutch Shell interest (La Corona Petroleum Co.) brought in its big well, which is estimated to have a capacity of 100,000 bbl. per day, in the Panuco field. During the spring of this year the activity of both the Federals and the Constitutionlists became very great in the vicinity of Tampico, culminating in a general exodus of the foreign employees of the oil companies in April, at the time of the Vera Cruz affair. Operations came to a standstill for a time, but the foreign employees gradually returned during the next two months, and restricted drilling was carried on by some of the larger companies, especially in the Panuco field. But continued guerrilla warfare, and the generally hostile attitude of all Mexican factions, led to heavy demands upon all companies, so that affairs in the oil fields were by no means normal when the European war broke out in August. In addition to this, lightning set fire to the big Potrero del Llano No. 4 well, which was not entirely extinguished until after the end of the year. During the latter part of the year hardly a string of tools was running. Gen. Carranza meanwhile appointed a commission to examine into the oil industry in Mexico and to make recommendations as to future methods of regulation. The conditions under which new work might be carried on under this decree were prohibitory, and resulted in a complete shutting down of all work during the early part of 1915. However, the Carranza government later modified its attitude, and agreed to issue permits to companies making application, under which they could operate under certain restrictions, pending the result of the work of the commission. This remains the situation in June, 1915, and development work is being carried on under difficulties, with only about six rigs running. Oil shipments have not been interrupted, however.

II. PRODUCTION

The following table shows the annual production of petroleum in the Mexican fields since 1904. Within the last three years, this has not represented the productive capacity, which was much greater. The completion of additional transportation lines to seaboard, the building of more tank ships, and the passing of the over-production of high-grade

oil in the United States, together with the broadening market following the end of the European trouble, will see the production of the Mexican fields advance enormously. At present they are yielding only about 70,000 bbl. per day; but the wells already drilled could produce 330,000 bbl. per day. In fact, the latest figures on good authority make this capacity more than 500,000 bbl.

PRODUCTION OF OIL IN MEXICO

Year	Barrels (42 gal.)	Year	Barrels (42 gal.)
1904	200,000	1910	4,099,000
1905	300,000	1911	13,655,488
1906	500,000	1912	16,844,066
1907	1,000,000	1913	24,574,500
1908	3,481,000	1914	25,725,403
1909	2,765,000		

During the latter part of 1914 there were 56 companies actually operating in the Mexican fields. But some 200 companies had been organized and were engaged in the business in one way or another. Most of these were American companies, the exceptions being the Pearson interests (English), the Dutch-Shell interest (English and Dutch), and a number of Mexican companies particularly in the Panuco and Topila field. The principal shippers of petroleum were the Mexican Eagle Oil Co., Ltd. (Pearson); the Huasteca Petroleum Co. (Doheny); the Tampico Company; the Mexican Gulf Oil Co. (Mellon); the East Coast Oil Co. (Southern Pacific Ry.); the Trans-continental Petroleum Co. (John Hays Hammond); La Corona Petroleum Co. (Dutch-Shell); the Panuco Valley Oilfields Corp., Ltd. (Simms & Bowser); and the Penn-Mex Fuel Co. (Standard Oil Co. of New Jersey).

Not more than 50 wells were actually producing, and more than half of the oil produced came from about six of these. The rest were shut in, waiting a better market, or better transportation, or for other reasons more or less connected with business policy, which sometimes included the obtaining of additional territory or the clearing up of titles.

The estimated total daily capacity of all wells in the Mexican fields of 330,000 bbl. may be divided among producing fields as follows, at the beginning of the present year:

Barrels	
150,000 Panuco	} Northern district, producing heavy oil, from 10 to 14° Bé.
4,000 Topila	
6,000 Ebano and Chijol	
65,000 Casiano and Los Naranjos	} Southern district, producing oil from 18 to 27° Bé.
100,000 Potrero, Alazan, Alamo, Tanguijo, and other points shipping from Tuxpam	

NOTE.—The Huasteca Petroleum Co.'s production at Cerro Azul and Juan Felipe has not been included through lack of information; but it is claimed that these wells are capable of adding 175,000 bbl. a day to this estimate, making the total 505,000 bbl.

While these figures indicate what might be expected of the Mexican fields if transportation and marketing facilities were to make it possible for all wells to be drawn upon to their full capacity, yet as a matter of fact, the year 1914 saw only an average of 70,000 bbl. per day of actual production distributed as follows:

	Barrels
Topila.....	398,679
Southern fields.....	18,830,359
Panuco.....	5,058,970
Chila-Salinas, Ebano, Chijol, etc.....	1,108,995
Tehuantepec.....	328,500
Total Production.....	25,725,403

This was consumed or stored as follows:

	Barrels
Clearances.....	20,674,357
Internal consumption.....	1,172,898
Refinery products.....	2,000,000
To storage.....	1,888,148

The addition to stocks brought the total storage in Mexico at the beginning of the year to 15,127,834 bbl. as follows: In steel storage, 9,658,258 bbl.; in earthen storage 5,369,576 bbl. (including 508,690 bbl. unmerch-
chantable).

Clearances from Mexican ports were distributed as follows during 1914:

	Barrels
To the United States.....	15,476,727
Mexican Coastwise.....	4,510,061
South America.....	195,138
To the Continent (Europe).....	356,205
To the United Kingdom.....	69,780
To Panama.....	46,446
To Cuba.....	20,000
	20,674,357

It is not to be expected that most of the producing districts in the Mexican fields will decline in a manner similar to the sand pools of the United States. It has been proved that the estimated initial capacity in many wells would be equaled if not surpassed by actual performance if they were allowed to flow freely. In fact, wells such as the Juan Casiano No. 6 and No. 7 and Los Naranjos No. 1 and No. 4, and the Mexican Fuel Oil Co.'s No. 3 Zurita, all in different districts, increased their production after being allowed to flow, as shown below:

	Initial Production, Barrels	Actual Flow Later, Barrels
Juan Casiano, No. 6.....	10,000	14,000
Juan Casiano, No. 7.....	20,000	28,000
Los Naranjos, No. 1.....	5,000	10,000
Los Naranjos, No. 4.....	10,000	50,000
Zurita, No. 3 (Panuco).....	30,000	80,000

These cannot be considered exceptional cases, since so few of the large wells have ever been allowed to flow freely. Considering geological conditions, which will be discussed later, such behavior is to be expected. The oldest commercial wells in the Mexican fields (Pez No. 1 in the Ebano district, and Juan Casiano No. 7) after producing since 1904 and 1910 respectively, and from widely separated districts, have shown practically no decrease in their flow since they were drilled in. The famous Potrero del Llano well, in the southern part of the field, has never produced up to its capacity and is still yielding as much as ever. These last two wells have produced approximately 40,000,000 bbl. of oil each during their history, and show practically no signs of decline, so far as known. The same may be said of the oldest well in the Panuco field (East Coast No. 401), drilled in 1910. This is even more remarkable, since the area surrounding this well has been closely drilled. While a few of the smaller wells at Panuco showed a decline after being shut in for a time, yet this apparently did not indicate any exhaustion, for at the same time No. 3 Zurita increased its production, as indicated above.

It must be said that producing conditions in Mexico differ fundamentally from those in the United States, and that, by reason of these conditions there is no vital need on the part of producing companies to bring the oil to the surface as quickly as possible (as at Cushing), and hence relatively little necessity for large tankage—except at terminal stations for operating reasons. The Mexican oil production comes from about 20 different distinct groups of wells. In all except Panuco and Topila, one company usually controls a large acreage surrounding its group of wells. Hence, each of these 20 districts represents a pool, or geological unit, in which only a few initial wells have been drilled. It is thus to the interest of each company to drill efficiently, and to keep wells shut in when there is no market or no transportation. There is usually no waste of oil or gas, except in the case of accidents, or inability to control a well. One of the few cases of waste happened recently, where a company struck a gas well south of the Topila field, which it is allowing to blow wide open, hoping it will show oil. This is almost the ideal condition advocated by some conservationists in this country—one company to each pool.

There has been little decline of pressure as yet, even in the older pools; and wells such as the first ones at Ebano and Juan Casiano, producing since 1904 and 1910 respectively, are doing so at practically their original rates. In the latter case 40,000,000 bbl. of oil have been taken from the one well, with not more than 10 per cent. decline from its initial daily production.

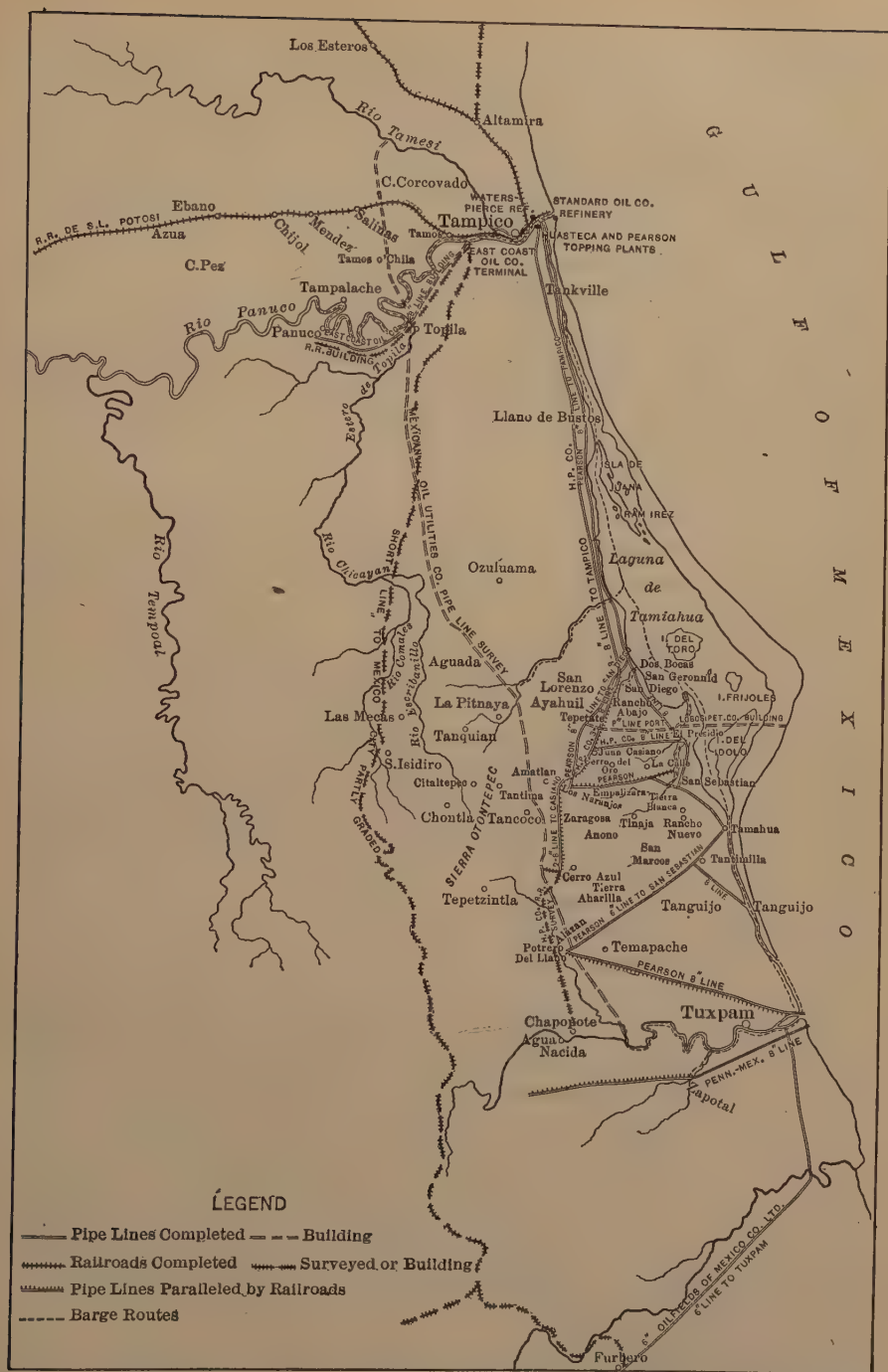


FIG. 1.—SKETCH MAP OF THE MEXICAN OIL FIELDS, SHOWING PIPE LINES AND RAILROADS.

Transportation.—Oil from the Ebano district is used largely as fuel by the Mexican railways, and considerable asphalt for paving also goes by rail. The Panuco oil all goes in barges to terminals near Tampico, and thence in tank ships to destination. The other fields all pipe their oil to seaboard either at Tuxpam or Tampico, where it is taken by tankers. There are at present 427 miles of pipe line in Mexico, mostly 8 in., some is 6 in. and smaller. About 50 miles additional are building or contemplated. The principal lines are as follows (see Fig. 1):

East Coast Oil Co.; 8-in. line, Panuco-Ponce; about 13.65 miles completed of a 24-mile line to Tampico.

Mexican Eagle Co. (Pearson); 8-in. line, Potrero del Llano to Tuxpam; 8-in. line, Potrero del Llano to Tierra Amarilla, to Tanguijo, to Tamiahua, to San Sebastian, to San Diego, to Bustos, to Tampico; 8-in. Los Naranjos to San Diego. (An allied company has a 6-in. line from Furbero to Tuxpam.)

Huasteca Petroleum Co. (Doheny); two 8-in. lines from Cerro Azul to Juan Casiano; three 8-in. lines from Juan Casiano to San Geronimo, to Esperanza, to Tankville, to Tampico; and a gas line from Juan Casiano to Tampico.

Penn-Mex Fuel Co. (South Penn Oil Co.); 8-in. line from Alamo and Agua Nacida to Tuxpam; 4-in. water line paralleling above.

Transportation is a serious difficulty in getting Mexican oil to the market. Present equipment could not possibly handle the full production of all wells hitherto drilled, which is variously estimated from 330,000 to 500,000 bbl. per day. In 1913, the production of these wells was 24,574,500 bbl. During 1914 none of the old wells declined in production, and 32 new wells came in, representing a possible new production of 196,000 bbl. per day. During the same year a number of pipe lines were enlarged from 6 in. to 8 in., and at least one new field (Los Naranjos) was connected with the seaboard. Shipping facilities from the Panuco field were increased, and some new tankers went into service. In spite of all this new production and new equipment, the field yielded only 25,725,403 bbl. of oil during 1914. At the present time not more than 50 miles of additional pipe line are under construction or contemplated, and very few new tankers,* considering this great shut-in production. This indicates that companies are awaiting a better market before making a serious effort to get their product from wells already drilled to points of consumption. A field with a possible daily production of at least 330,000 bbl., which meanwhile only produces 70,000 bbl. per day, and at the same time makes little effort to build new pipe lines and tankers, as indicated by present constructions under way, furnishes strong evidence of the effect of market conditions.

* Since writing the above, the author is informed that one large company has contracted for the building of a large number of additional tank-ships.

It is to be assumed that development in Mexico will be relatively much slower than would be the case if these fields were located within the United States, for the following reasons:

1. The great expense of operating keeps out all except financially strong companies which will not develop their territory so rapidly as to break the market, and, since they control large areas around their wells they can afford to suspend production, awaiting a better market, because they are not losing pressure or oil to their neighbors in the meantime.

2. Political difficulties retard development.

3. Government and State taxes, high operating and maintenance charges, as well as the cost of transportation by tank ships, add to the cost of the oil delivered to markets such as the United States. These expenses make it impossible to put oil into United States markets for less than 35 c. per barrel at Gulf Coast ports or approximately 50 c. for Atlantic ports. The price for this oil cannot break below these costs for any except small lots, offered at times of a crisis in the field. This is particularly true, since the largest producers also control, either in Mexico or in the United States, their own pipe lines, tankers and refineries.

It may be added that Mexican oil at those prices, plus a profit, must compete with Gulf Coast and California fuel oil, and Oklahoma residues. The fact that it cannot be brought to this country and sold at a price much below these domestic oils prevents its being diverted into as many new channels as usually open when the price breaks to a low point; hence the market will not expand so rapidly. In so far as transportation is a factor of this relatively high price for Mexican oil, it is the reason for this restricted market. And it must be admitted that transportation is thus indirectly the chief factor in keeping down production.

The application of fuel oil to marine uses, and its extension for many purposes at seaboard points, especially in the East, is urged. The development of this market has been very backward, when the lower cost and greater efficiency over coal at most seaboard points are considered. Now that a sufficient supply is in sight for the future, it will be to the advantage of one of the large producing and marketing companies in the Mexican field to develop the possibilities of the extended use of fuel oil in various types of engines and boilers, in and around New York harbor.

III. OPERATIONS IN DETAIL

The following was the approximate status of all operations in the Mexican fields during the spring of 1915:

MEXICAN EAGLE PETROLEUM COMPANY, LTD. (Pearson interest)

Well No.	Property	Daily Production Barrels	Remarks
1, 2, 3 & 4...	Dos Bocas (Lease) Camp abandoned	Small 200,000	Nos. 1, 2, & 3 were small producers. No. 4 came in at 200,000 bbl. a day and burned for 57 days, ruining the field; 1,824 ft. deep
1, 2 & 3.....	San Antonio de Tamihui (Lease)	Dry holes, all more t h a n 3,000 ft. deep
.....	Potrero del Llano (Lease)	Wells about 2,000 ft. deep
1.....	800	Shut in
2.....	600	Shut in
3.....	Dry hole
4.....	110,000	Gusher
5.....	Shut down
6.....	Abandoned
7 & 8.....	Drilling
.....	Alazan R a n c h (Lease)	Adjoining Potrero del Llano
1 & 5.....	Abandoned
2 & 3.....	Salt water
4.....	11,000	Shut in
6.....	3,000	Shut in
7.....	2,000	Shut in
8.....	Crooked hole
9.....	3,000	Shut in
10.....	1,000	Shut in
11, 12 & 13	Drilling
.....	Tierra Amarilla (Lease)	South Field
1 to 5.....	Salt water and small amount of oil
6.....	Can produce 6,000 bbl. oil and 4,000 salt water	Shut in
7, 8, 9, & 10..	Producing salt water and small amount of oil.
11 to 15.....	Drilling or derricks
16.....	Location

MEXICAN EAGLE PETROLEUM COMPANY, LTD. (Pearson interest).—*Continued*

Well No.	Property	Daily Production Barrels	Remarks
..... 1 to 20.....	Tanguijo (Lease)		South field Total production about 500 bbl. a day. Nos. 2, 5, 6, 7, 8, 9, 10, 11, 13, 15, 17, and 18 are shallow wells about 1,300 ft. deep. No. 20, 1,750 ft., 20 gravity oil, came in Mar. 11, 1914, located 3 miles from Nos. 1 to 19 inc.
..... 1..... 2 & 3..... 4..... 5, 6, & 7.....	Naranjos (Lease and fee) 10,000 50,000	Amatlan 20 gravity oil 1,885 ft. deep Dry holes, about 2,300 ft. Flowed this amount for a week when gate valve broke Locations
..... 1 & 3..... 2.....	San M a r c o s (Lease) 150	South field, 20 gravity Dry at about 2,800 ft. Shut in
..... 1..... 2.....	Tierra B l a n c a (Lease) 20	35 gravity oil Abandoned Shut in; 3,100 ft. deep
..... 1 & 2..... 3 & 4.....	Tlacolula (Lease) Dry holes at 4,100 ft. Drilling and derrick
..... 1 & 2..... 3 & 4.....	San Pedro (Lease) 10 (each)	State San Luis Potosi Shut in; 45 gravity oil Drilling

HUASTECA PETROLEUM COMPANY (Doheny Interest)

.....	Juan C a s i a n o (Fee and lease)	Chinampa, 19 gravity
1.....	400	Shut in; 1,700 ft.
2.....	100	Shut in; 1,700 ft.
3, 4 & 5.....	Abandoned
6.....	10,000	Shut in; 2,363 ft. deep
7.....	25,000	Has been flowing 5 yr.; 2,112 ft. deep
8 & 9.....	Drilling

HUASTECA PETROLEUM COMPANY (DOHENY INTEREST).—(Continued)

Well No.	Property	Daily Production	Remarks
.....	Cerro Azul (fee)	South field
.....	Reported 27 gravity oil
1	Abandoned
2 to 5 inc.	Shut in
.....	Juan Felipe (fee)	South field
1	Shut in
.....	Five wells are reported completed on these two properties, with a combined capacity of 175,000 bbl. daily. Pressures reported from 575 to 1,080 lb. for different wells	
.....	Pithaya (fee)	Northwest of Casiano
1 & 2	Abandoned
3 & 4	Drilling
.....	Juarez (sub-lease)	Adjoining Pithaya
1 & 2	Show in No. 1	Shut down
.....	Ebano field	Fifty miles west of Tampico
1 to 27	8,000	All but 5 shut in; 11 gravity oil

PENN-MEX FUEL COMPANY (SOUTH PENN OIL Co.)

.....	Alamo Ranch (Lease)	South of Tuxpam River
1	2,000
2	20,000
3 & 4	Drilling
.....	Agua Nacida (Lease)	South of Tuxpam River
1	Show
2	Abandoned 2,700 ft.
3	Drilling
4	Derrick
.....	Tamatoca (Lease)	Near Potrero del Llano
1	Drilling at 2,950 ft. last year
.....	Las Canas (Lease)	South of Tuxpam River
1	Drilling at 3,000 ft.
.....	Molino (Lease)
1	Derrick

INTERNATIONAL PETROLEUM COMPANY (JOHN HAYS HAMMOND INTERESTS)

Well No.	Property	Daily Production	Remarks
..... 1 & 2.....	Tinaja Ranch	South of Amatlan Dry holes
1..... 2.....	San Marcos	Near Tanguijo, shut down at 3,200 ft. Derrick

MEXICAN EASTERN OIL COMPANY

..... No. 1.....	San Diego Ranch (Near Dos Bocas) Abandoned at 2,785 ft.
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ELECTRA OIL COMPANY

..... 1.....	Rancho Abajo. (East of Casiano) Show oil Drilling
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MEXICAN PREMIER OIL COMPANY (SPELLACY)

..... 1.....	Soledad Ranch. (north of Cuecil- los) Shut down at 2,350 ft.
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HIDALGO PETROLEUM CO. (M. A. SPELLACY ET AL)

..... 1.....	La Calle Rigging up
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CORTEZ OIL COMPANY (AMERICAN TOBACCO CO. INTERESTS)
(Now the Port Lobos Petroleum Co.)

3..... 4..... 1.....	Empalizada (Tepetate) (north of Casiano) Sandoval 25,000 4,000	Shut down at 2,700 ft. Shut in. Casing trouble (Offsetting No. 4 Tepetate)
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GALEY ET AL

Well No.	Property	Daily Production	Remarks
.....	San Diego de la Mar.....	In Chinampa
1.....	Shut down at 2,100 ft.

STANDARD OIL COMPANY OF MEXICO

.....	El Gallo Ranch (Lease)
1.....	Abandoned at 2,727 ft.

MEXICAN FUEL OIL COMPANY

.....	Caracol Ranch (11 to 12 gravity oil)	West of Tampico
1.....	Abandoned, shallow
2, 3 & 7.....	350	Shut in, shallow
4, 5 & 9.....	Shut down
8.....	175	Flowing, 770 ft. deep

TAMPICO FRUIT COMPANY

.....	Caracol Ranch	Fifteen miles up the Panuco River
1.....	2	Shut in 2,910 ft.

TAMPICO OIL Co., LTD.

.....	Chila Ranch	Near Caracol; 12 Gravity
1 & 4.....	205	Flowing
2 & 3.....	50	Shut in
.....	Salinas Ranch	16 gravity oil
2.....	100	Sanded up; 1,603 ft.
3.....	Drilling

DOS BANDERAS OIL & GAS Co.

1.....	Dos Banderas lease (Near Tampico)	Small production	Shut down; 3,480 ft. deep
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SMITH OIL Co.

No. 1.....	Caracol lease	Drilling at 1,925 ft.
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Topila Field

Company	Wells, Completed	Wells, Drilling and Locations	Wells, Dry or Abandoned	Estimated Daily Pro- duction, Barrels
Mexican Fuel Co. (Pierce Oil Corp.)	4	1	1	750
East Coast Oil Co. (Southern Pacific R.R.).....	3	3	1	100
Mexican Oil Co.....	1	1	2,000
American Fuel Oil Co.....	1	5,000
La Corona (Dutch Shell).....	2	1	1	130
Mexican Gulf Oil Co. (Mellon interests).....	1	1
Tampico Panuco Oil fields.....	1	2	1
Topila Petroleum Co. (Spellacy)...	1	1
Penn-Mex Fuel Co. (South Penn)	1	1
Scottish Mexican Oil Co.....	1	1
M. C. Anderson.....	2	52
W. H. Milliken.....	1	20
R. L. Brooks (Producers Oil Co.)	1	450
Cia. Petrolera de Monterey	1	2	105
Mexican Associated Oil Co.....	1	1
Cia. Explotadora de Topila	1	1

Panuco Field

Company	Wells, Completed	Wells, Drilling and Locations	Wells, Dry or Abandoned	Estimated Daily Pro- duction, Barrels
Vera Cruz Mexican Oil Syndicate, Ltd.....	5	1	10,000
International Petroleum Co.....	4	4	2	800
Transcontinental Petroleum Co....	1	1	800
Gulf Coast Corp.....	3	1	100
Mexican Development Co.....	1	1
La Corona Petroleum Co.....	4	2	1	100,600
Freeport and Mex. Fuel Oil Corpor- ation.....	5	4	1	31,500
National Oil Co.....	9	3	1	21,000
Mexican National Oil Co.....	2	2	1
Tampascas Oil Co.....	2	1	1	350
Penn-Mex Fuel Co.....	1	1	25
Piedras Development Co.....	1	25
Cia. Petrolera Las Brujas.....	2
Producers Oil Co. of Tex.....	2	6,500
Cia. Petrolifers, S. A.....	2
English Oil Co. of N. Y.....	1	3	1
Cia. Petroleo Mexicana.....	1	1	75
Excelsior-Panuco Oil Co.....	1	1,000
Mexican Gulf Oil Co.....	4	1	1	12,000
Scottish Mexican Pet. Co.....	1	250
M. A. Spellacy.....	1	1,000
East Coast Oil Co.....	7	3	8,000
Mexican Development Co.....
Tal Vez Oil Co.....	1	1	1,500
Victoria Oil Co.....	3
Heradura Oil Co.....	1	1
R. E. Brooks.....	1
Cia. Petroleo Maritima.....	1
La Bonanza Pet. Co.....	1	1	200
New Bonanza Oil Co.....	1
Cia. Abastecedora de Petroleo y Aceite.....	2
Yisarri & Blanco.....	1
Cia. Petroleo Mexicana La Nacional	1
Cia. Petroleo Los Perforadores.....	1
Cia. Indio de Petroleo.....	1
Los Dos Estrellas Oil Co.....	1
Cia. Petroleo La Oxaquena.....	1
Cia. Petroleo Panuco-Manguaubes	2
George Harmon.....	1	80,000
Alamo Oil Co.....	1
Cia. Hispania-Mexicana.....	1
Panuco-Topila Pet. Co.....	1
El Vado Petroleum Co.....	2
Panuco Valley Oil Fields Ltd.....	1	2,000
Pan-American Oil Co.....	1
Fenix Oil Co.....	1

Summarizing the above, we find that there were in the Mexican fields during the spring of 1915, 53 wells producing, 111 shut in, and 40 abandoned, while the number of wells drilling, shut down or simply located, was 105.

1. *Markets*

All oil exported from Mexico goes by tank ships. During normal times most of the Mexican railroads are equipped for oil burning, and obtain their fuel from the Ebano field, or from the several refineries at Tampico or Miniatitlan after it has been "topped." By far the greater proportion of all Mexican oil which is exported goes to United States ports, where it is "topped" and the residue is sold as fuel or for paving purposes. During 1914, shipments of Mexican oil were as follows:

Destination	Barrels (42 Gal.)
United States.....	15,476,727
Mexican Coastwise.....	4,510,061
South America.....	195,138
Continental Europe.....	365,205
United Kingdom.....	67,780
Panama.....	46,446
Cuba.....	20,000
Total clearances.....	20,674,357

Refineries.—There are in Mexico the following refineries which supply the domestic market with refined oils:

	Daily Capacity, Barrels
Standard Oil Co. of New Jersey at Tampico...	4,000
Waters-Pierce.....	Small
Pearson (Mex. Eagle) (Eventually 25,000), Tampico.....	12,500
Pearson (Mex. Eagle) at Minatitlan.....	15,000 to 25,000
Huasteca Pet. Co. at Tampico.....	10,000
Huasteca Pet. Co. at Ebano Asphalt Plant....

2. *Quality*

The oil produced in these fields is divided roughly as to quality into two classes—one heavy gravity, running from 10 to 14° Bé (0.993 to 0.973 sp. gr.); and one relatively lighter, running from 18 to 22° Bé (0.947 to 0.922 sp. gr.). A small quantity of very light crude oil, of 30 to 40° Bé. (0.840 sp. gr.), has been discovered, but no promising developments of it are assured.

The distillation analysis of this oil yields the following percentage of refined-oil products, compared with Illinois and Oklahoma crude oil:

	Yields of Finished Products from Mexican Crude Gravity 25° B _é Per Cent.	Yields of Finished Products from Illinois and Oklahoma Crude, Process No. 3, Per Cent.
Benzine.....	7.14	21.00
W. W. Burning Oil.....	21.42	18.38
Gas oil.....	21.42	28.81
Neutral Distillate for Red and Miscel- laneous Lubricating Oil.....	11.71	7.68
Asphalt.....	32.16	17.62
Process Loss.....	6.15	6.51
	<hr/> 100.00	<hr/> 100.00

And the following analyses of characteristic components and heat value are reported by Messrs. Dow & Smith, Chemical Engineers, New York City:

REPORT OF ANALYSIS NO. 6804, JUNE 11, 1912.

Specific gravity at 60° F.....	0.9115
Equivalent to.....	23.8°B _é
Flash, N. Y. State open tester.....	77° F.
Flash, N. Y. State closed tester.....	65° F.
Fire test.....	120° F.
Boiling point (distillation begins).....	175° F.
Water.....	Trace
Viscosity at 32° F. (Engler 1st 50 c.c.), sec.....	1,980
Viscosity at 72° F. (Engler 1st c.c.), sec.....	298
Loss on heating 20 g. 5 hr. at 325° F., per cent.....	26.8
Fixed carbon, per cent.....	7.23
Paraffine scale, per cent.....	2.25
"Asphalt Contents" 50 pen. (slow evaporation 32 hr. in air bath at 300 to 400° F.), per cent.....	59.5
"Asphalt Contents" 50 pen. (quick evaporation by burning), per cent.....	52.0
B.t.u.....	18,493

Ultimate Analysis

	Per Cent.
Carbon.....	82.83
Hydrogen.....	12.19
Oxygen (by difference).....	0.43
Nitrogen.....	1.72
Sulphur.....	2.83

REPORT OF ANALYSIS, NO. 7288—CRUDE MEXICAN OIL

Specific gravity at 60° F.....	0.985
Equivalent to.....	12.1°B _é
Flash (closed tester).....	157° F.
Fire test.....	225° F.
Sulphur, per cent.....	4.20
Scale paraffine, per cent.....	0.518

Calorific value.....	18,227 B.t.u.
Viscosity at 60° F.....	Too thick to take
Viscosity at 100° F.....	Too thick to take
Viscosity at 212° F. (200 c.c. Engler Viscometer)...	615 sec.
Flows readily at.....	150° F.

Fractional Distillation

	Per Cent.	Products,
1st drop, 140° F.		
140 to 302° F.....	4	naphtha distillate
302 to 545° F.....	20	burning oil distillate
545 to 600° F.....	10	cracking commenced
600 to 620° F.....	15	cracking continued
Residue and loss.....	50	cracking badly, temperature fell to 605° F.
Penetration at 77° F. of residue from fractional distillation.....	60	

Asphalt Contents by Evaporation

100 penetration at 77° F., per cent.....	70.1
Ductility of ditto, cm.....	100.0
50 penetration at 77° F., per cent.....	65.7
Ductility of ditto, cm.....	100.0

The following are analyses of the Huasteca oil:

HUASTECA OIL,

CASIANO FIELD, WELL No. 7,

SAMPLE I

Gravity 19.4° Bé.....	Pour, 0
Flash, 80° F.....	B. S., 3 per cent.
Viscosity, 450 B. O. T.....	B.t.u. per pound, 19,124

Fractional Distillation

Temperature	Per Cent.	Gravity, Degrees Baumé
Over from 259 to 385° F.....	10	58.7
Over from 385 to 493° F.....	10
Over from 493 to 500° F.....	1	45.9
Total		
Distillates.....	21	51.5
Residue.....	78	13.5; pour, 40
Loss.....	1	
	100	

SAMPLE II

Gravity 19.3° Bé.....	Pour, 0
Flash, 96.....	B. S., 2 per cent.
Viscosity, 500 B. O. T.....	B.t.u., per pound, 19,160

Fractional Distillation

Temperature	Per Cent.	Gravity, Degrees Baumé
Over from 244 to 400° F.....	10.0	57.8
Over from 400 to 493° F.....	10.0
Over from 493 to 500° F.....	0.6	45.1
Total:		
Distillates.....	20.6	50.8
Residue.....	78.5	13.4
Loss.....	0.9	
	<hr/> 100.00	

3. Costs and Prices

The cost of production of oil (of about 12° Bé.) at Panuco at present is about as follows:

Cost per Barrel of 42 Gal.	U. S. Gold, Cents
Bar dues (*50c. Mex. per Metric ton or 6.6 bbl. \pm av.)	3.8
Production tax (30c. U. S. per Metric ton).....	4.5
One-tenth Royalty (based on an arbitrary value of 50c. U. S. per barrel).....	5.0
Transportation by barge, Panuco to Tampico.....	6.0
Lighterage (terminal to shipside at Tampico).....	2.0
Production expenses (estimated).....	10.0
	<hr/>
Cost of barrel of oil, f.o.b. ship at Tampico.....	31.3

(*NOTE: This is Mexican gold with an exchange value of 2:1).

The cost of transportation by tank ship to United States ports may be estimated as 10c. per ton per day. As the voyage consumes from 3 to 10 days, depending upon conditions and the port to which the oil is consigned, the total cost to destination will vary from 35.5 to 45.3c. per barrel. Costs from Tuxpam will differ with differing producing conditions in the field; pipe line instead of barge transportation, and no bar dues; but with a slightly longer sea voyage for tankers.

The official market price during 1914 for Panuco heavy oil f.o.b. ship at Tampico, ranged from 40c. a barrel in January to 20c. (U. S. gold) at the close of the year. Evidently some oil was sold at less than the cost of production, in order to keep equipment in use, and to take care of production which could not be shut in. Very little oil is sold at the wells in the Mexican fields, although a few sales have been reported at Topila and Panuco.

The lighter oils (19 to 27° Bé.) from the southern districts, are all sold under contract at higher prices than the Panuco oil. Casiano oil is valued at from 50 to 60c. f.o.b. ships at Tampico, which would make its value at United States ports from 55 to 75c. per barrel. The oil from the properties of the same company at Jan Felipe and Cerro Azul is reported to be as light as 27° Bé. The oil from the Alamo lease of the Penn-Mex Fuel Company is about 24° Bé., which is slightly higher than that from the Potrero del Lano lease of the Mexican Eagle Petroleum Co.

4. *Drilling*

Drilling is carried on in these fields with heavy combination rigs, usually of the California type. The rotary is used through the soft upper marls (Mendez or Los Esteros beds) into the upper portion of the San Felipe formation. Here the casing is set on top of some hard limestone shell, and drilling is continued with percussion tools through the San Felipe and well into the massive Tamasopa lime, unless oil is found sooner. In most of the large wells the oil flow has been struck in broken blue shale or limestone, a short distance above the top of the Tamasopa. In others, the oil is found in the upper few feet of the Tamasopa. There are no defined oil "pays" which can be correlated with each other from one field to another, or even from one well to another in the same field. The oil occurs in fractured zones, in which solution channels caused by circulating underground waters are frequent. This condition has led to the generalization, based on the experience of operators in these fields, that all the big wells get their oil "high up;" or in other words at a shallower depth than the average for the locality.

Drilling is very expensive in these fields. An operator expecting to drill must estimate that his first wells, if wild-cats, will cost in the neighborhood of \$65,000 each, exclusive of the leasing expenses. This excess outlay may be divided under the heads of high wages; the establishment of camp and commissary; the transportation and duties on drilling materials; the delays due to accidents and waiting for materials, or the cost of keeping a large stock of extra parts to prevent such delays; road building and rights-of-way for roads and water lines; the high cost of fuel, for wild-cat wells; and the cost of keeping up communication with Tampico, by means of launches and horses.

The difficulties and expenses of drilling are affected by geological conditions also. Along the coast line, and the shores of Lake Tamiahua, there are deposits of varying thickness, and of Quarternary and Recent age. This is also true in the Panuco field, where they are about 100 ft. thick, and obscure all evidence of structure and intrusion. The Mendez beds vary considerably in thickness within the same district; and the same appears to be true of the San Felipe series. There are no horizons within either of these formations which can be correlated, even between two nearby wells. And owing to the difficulty in determining exactly from a well record the top of the Tamasopa limestone, due to the gradual transition from the San Felipe series to a massive limestone (Tamasopa) phase near the bottom, it is impossible to ascertain positively the extent and position of such unconformities as probably exist. This is even more difficult because of the lack of coöperation upon the part of the various companies doing geological work in these fields. The isolated location of the various districts permits a degree of secrecy possible in

find a purchaser; or promote a company of his own to handle the well; or enter into an alliance with others who will furnish money to build pipe lines to seaboard, erect a terminal, charter or build tank ships, and finally market the oil in the United States or elsewhere. As matters stand at present, with so many wells shut in, the finding of the oil is the least important step, particularly if there is not enough territory leased in the vicinity to protect such a well, and justify capitalists in undertaking the further development of the project.

IV. GEOLOGY

1. *Stratigraphy*

In the Mexican fields four distinct formations are encountered: (1) an upper series of fossiliferous Tertiary sandy limes and sandstones, interbedded with limy and sandy clays, the beds varying in thickness from 600 to 1,300 ft.; (2) an intermediate section, 2,000 to 3,500 ft. thick, of gray marls and shales (called Mendez marls, or Los Esteros beds), the upper portion of which has been identified as Eocene Tertiary which give place in their lower portion to (3) the San Felipe or Valles beds of limestone shells alternating with blue and brown shales 200 to 700 ft. thick. These lie upon (4) a massive blue-gray (Tamasopa) limestone formation, at least 3,000 ft. thick, fossiliferous in its upper portion, of Lower Cretaceous age. In some areas, the upper Tertiary limestones have been eroded away, and the surface formation consists of marls (see Fig. 2).

a. The Upper Tertiary.—The upper Tertiary formations have been divided by some geologists into the western or Tanlajas series, and the eastern or Ozuluama and Temapache series. While they keep their general character as marine deposits of rapidly alternating sandy limestones and clays, and local beds of fossils such as the beds of nummulites and orbitoides in the central districts can be correlated for short distances, yet such correlations cannot be carried from one area into another without much care.

In the western or Tanlajas district these beds average about 1,100 ft. in thickness, while in the easternmost wells—those at Tanguijo—they are at least 1,200 ft. thick. These upper Tertiaries, however, practically cover the southern and central portions of the Mexican field, except where river valleys have exposed the underlying marls by erosion, or where large basaltic intrusions have taken their place.

These upper Tertiaries have been found oil-bearing at one locality only—Tanguijo. Here the Aguila company has drilled a number of wells, which are capable of pumping about 500 bbl. per day. The oil is 21°Bé. and the wells decline rapidly. Near the bottom of the upper Tertiary section in the Tuxpam district, a dark bituminous sand bed is reported. This does not appear in the northern section. However, owing to the

friability of these beds, and the prevalence of strong jointing, one finds a great many asphaltic seepages exuding from them at the surface. In most cases this is due to the fracturing of the underlying beds, and the intrusion of dikes; the oil finding an easy egress through these upper beds. It is believed by some that the oil found in these beds at Tanguijo has found its way there through migration from lower formations, through fractures. I regard this as very probable. The size of these wells and their behavior hardly encourages prospecting in these beds for the development of commercial production, especially while wells in other districts are so prolific.

While there are numerous exposures of these upper Tertiaries in the central and southern districts, and good clinometer readings can be obtained at many points, yet such data frequently prove misleading. The absence of good key horizons makes it difficult to locate fault lines, or to determine their throw. It is also frequently impossible to work out the details of the monocline structure found in these beds; but since these minor irregularities are probably of little importance, this difficulty is not of great consequence. In other words, intrusions and fractures, channelling in the oil-bearing limestone, and the effect of sedimentary gradients and unconformities are all of so much greater importance in these fields in their influence on oil accumulation, that only the large well-marked structural features need be considered, where these other factors are not known to exist.

b. The Mendez Marls.—The Mendez marls consist of a very uniform deposit of gray to blue shales and marls. In regions of steep folding, these often show bold jointing near the surface. There is practically no change in their lithological character from top to bottom. They average from 2,000 to 3,500 ft. in thickness. In the Huasteca country, they are nearer the former thickness. A few irregular beds of sandy limestone are reported in this formation, but they are not persistent. However, there are frequently small quantities of heavy oil immediately below them, although never enough to sustain production of commercial importance.

The apparent thickening or thinning of these beds within short distances is sometimes accounted for by faulting of the formations. Or a local steepening of the dip may cause an apparent thickening. The upper limestone lenses in the San Felipe beds likewise come in and go out, and seem to indicate unconformity; so that the thickening of the Mendez marl beds is counterbalanced by a smaller thickness of the lower San Felipe series, the depth to the top of the Tamasopa limestone remaining practically constant. Even this interval, however, is not always constant in the same district, for the reason that there is more or less unconformity between the top of the Tamasopa and the overlying San Felipe shells.

c. *The San Felipe Formation*.—This may be described as a transition series between the upper Mendez marls and shales and the underlying massive Tamasopa limestone. It begins with an occasional thin limestone shell. These increase with depth in number and thickness, being interbedded with blue shales which conversely decrease in thickness downward until the series gives place to massive limestone. These beds apparently vary in thickness from about 300 to as much as 800 ft. Since drillers frequently do not begin to record these shells until they are about ready to set the casing, it is impossible to estimate the thickness of the series in every well merely by the well log.

These San Felipe beds are frequently the oil-bearing formation in the Mexican fields, particularly for the largest wells. This is because of the well-known occurrence of strong fracturing, with which large oil accumulations are associated. Oil is found in such a fracture as high as it can go; and any well drilling into such a "fissure" will encounter its big flow "high up," in the terms of the operator; the marls finally proving an effective seal against its escape above a reservoir in the San Felipe. Among others, the following "gushers" never reached the Tamasopa lime: Juan Casiano No. 6 and No. 7; Pearson's Los Naranjos No. 1 and No. 4; La Corona No. 5 at Panuco, and Spellacy No. 1 at Panuco. In such cases, the oil is usually found in broken shale or channelled limestone in the San Felipe formation. However, since this channelling is probably more frequent at the base of the series, where there is considerable unconformity, the top of the Tamasopa lime has come to be looked upon as the "pay" formation.

While such faulting or fracturing extends to the surface through the Mendez beds, as indicated by dikes and seepages, yet it is perfectly evident that as these upper formations contain no porous beds, no oil accumulations can be looked for above the harder San Felipe. As these soft marls have undoubtedly been pinched together after disturbance took place, they have in many instances effectually sealed any outlet except such passageways as those kept open by igneous intrusions. However, such seepages may travel laterally for considerable distances along joint cracks in the harder upper Tertiaries, before finding an outlet to the surface—making the successful location of wells problematic if based upon no other evidence than a seepage of *chapopote*.

d. *The Tamasopa Limestone*.—Jeffreys says that the limestones of the San Felipe series can be distinguished from the lower Tamasopa lime in that (1) they contain no fossils; (2) they are not massive; and (3) their texture is generally of more even grain than that of the Tamasopa. These differences, however, are not always easy to detect in a well sample. The Tamasopa is a compact, hard, gray, crystalline limestone, fossiliferous near the top. It has never been drilled through, and no productive horizon is known far below the top. There are no oil seepages or evi-

dences of hydrocarbon contents at the outcrop, far from the contact of overlying formations. There appears as yet no conclusive evidence that the oil in the Mexican fields had its origin in the beds of the massive Tamasopa limestone, as claimed by some geologists. The very nature of such beds, especially considering the subsequent period of erosion, is against such an assumption. On the contrary, there is more evidence here than in many other fields pointing to the probability of its having migrated from the overlying marine marls. This will be discussed in another paragraph.

2. *Structure*

All the large wells drilled up to the present are located where there exists a significant combination of both favorable anticlinal or dome structure with pronounced fracturing of the formations. These fractures (frequently faults of relatively small throw) are usually accompanied by basaltic intrusions, and seepages of asphalt and gas. In the Panuco field, all surface evidence both of structure and intrusion, has been obscured (except at one point north of the river in the Tampalache area) by about 100 ft. of alluvial sediments; but drilling has shown that the same conditions exist and have influenced the oil accumulation here as in other parts of the Mexican fields (see Figs. 3 and 4).

There has also been faulting in connection with some of these anticlinal structures, especially those nearer the mountains (Fig. 5). These folds become broader and less frequent to the eastward (see Fig. 5). However, other forces caused a number of relatively well-marked folds in the vicinity of Otontepec, such as those at Potrero del Llano and Los Naranjos. This folding was caused by lateral thrust and probably certain vertical stresses incidental to the formation of the Sierra Madre mountains to the west. These in turn set up other lines of weakness in the formations, through which during late Tertiary time other igneous rocks were intruded. This is shown by Fig. 6, which is a map of the central part of the fields in which many of the main basalt dikes have been located by the author.

A study of this map will reveal a number of interesting relations, for instance, the general agreement between the strike of the sedimentary formations and that of the main dikes in the coastal portion of the fields. A reference to the sketch map (Fig. 2) showing the general areal geology of the Mexican field, fails to reveal any locality where the Tamasopa lime or even the San Felipe beds have been thrust up to the surface by intrusives, as claimed by some of the earlier writers. The writer knows of no instance where any pronounced doming has been caused by the upthrust of dikes or so-called "plugs" of basalt. Some very local distortion and faulting has been caused at certain places; but such instances are balanced by others in which the sedimentaries actually dip toward



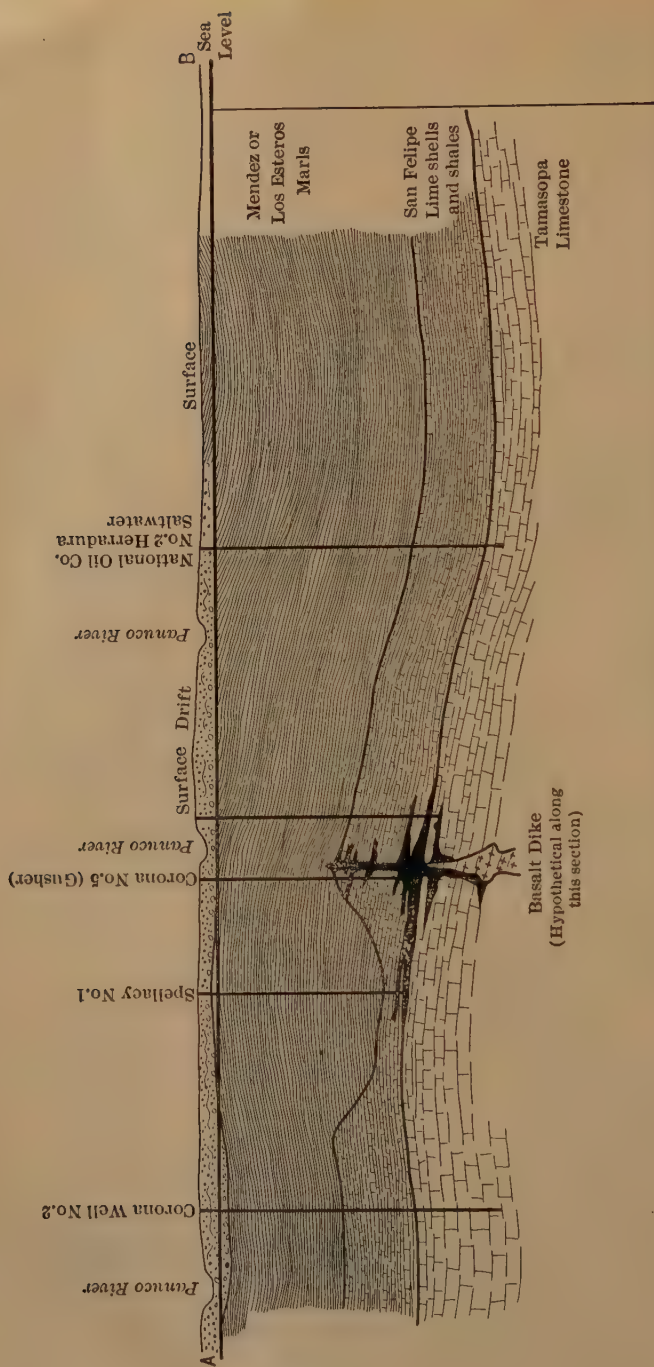


FIG. 4.—HYPOTHETICAL SECTION THROUGH THE PANUCO FIELD, MEXICO, ALONG THE LINE A-B (FIG. 3), SHOWING ANTICLINAL TERRACE AND FRACTURE.

(Note: The character of the unconformity as shown between the marls and the San Felipe beds is tentative.)

large igneous bodies from all sides. That is to say, shrinkage due to cooling has more than offset any "doming" effect.

In common with other fields where the surface beds are friable, the local dip of such beds as measured by a clinometer is apt to be much greater (especially near intrusives) than that of the underlying formations. Additional drilling has failed to corroborate the "nail" theory of the effect of igneous plugs or other intrusions upon the structure of surrounding sedimentaries, in the Mexican field.

A reference to Fig. 7 will show that some secondary folding by lateral thrust has been caused by the intrusion of the Sierra Otontepec—the largest igneous mass in the field west of the mountains.

Igneous Intrusions.—Again referring to Fig. 6, it will be seen that the fields at Juan Casiano, Los Naranjos, Dos Bocas and Panuco are all located at the intersection of strong fractures, where such intersections occur on anticlinal folds. Access to the well records of certain operating companies would undoubtedly reveal interesting data with regard to the relation of the accumulation of oil under these conditions to possible fault displacement. Where the marls are exposed at the surface, as at



FIG. 5.—GENERALIZED SECTION EAST AND WEST THROUGH NORTHERN PART OF THE OIL FIELDS. AFTER JEFFREYS.

Los Naranjos and Dos Bocas, it is impossible to determine local faults from surface indications, unless they have considerable throw or linear extent, enough to make it possible to locate them by inference. Nor is it at all easy to determine the throw of such possible faults, even where the harder Tertiary beds are found at the surface, due to the lack of well-defined key horizons. However, sufficiently detailed work will usually give enough clues to their existence, especially after a knowledge of conditions in the surrounding areas has been gained.

Intersections of strong fractures are frequently accompanied and marked at the surface by conical basalt peaks, which usually represent the "mushrooming" of an igneous neck intrusion. Wells drilled close to the contact at several of these conical hills have disproved the theory, advanced by one geologist, that they were "plugs" whose conical shape persisted at depth. Such wells have frequently started close to the contact and have been drilled into the oil formation at more than 2,000 ft. in depth, without encountering any further basalt, or any violent distortion. These intrusions and accompanying flows apparently follow

some fracture intersection where the resistance was least, and merely represent knots along the line of a dike (Fig. 6). At other places cone-shaped hills occur, also along the line of some fracture or dike-line pro-

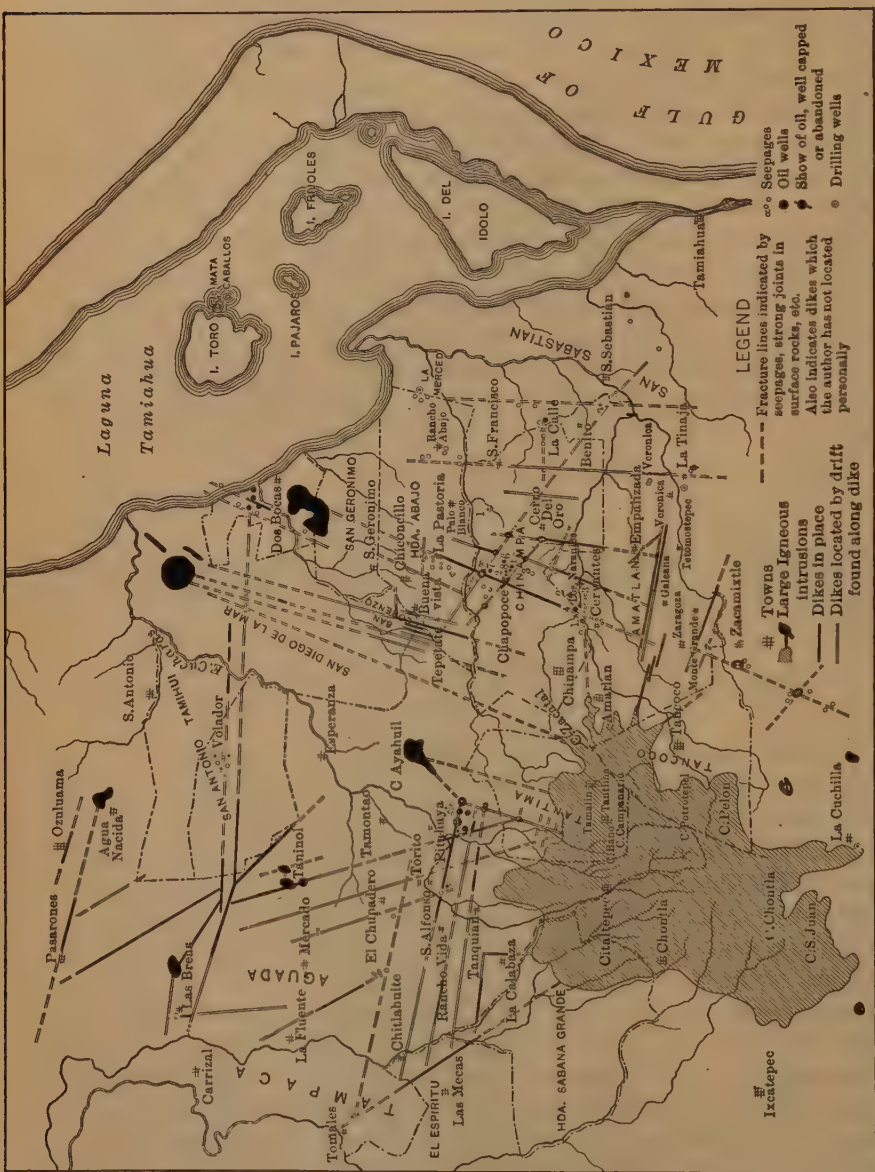


FIG. 6.—MAP OF A PORTION OF THE MEXICAN OIL FIELD, SHOWING THE RELATION OF THE PRINCIPAL IGNEOUS INTRUSIONS TO OIL ACCUMULATION AS EVIDENCED BY SEEPAGES AND LARGE GUSHERS WHICH HAVE BEEN DRILLED AT DOS BOCAS, JUAN CASIANO, AND LOS NARANJOS.

jected, but with no sign of basalt at the surface, and no evidence of violent folding. The formations at such places seem to be in place, yet considerably harder than the surrounding district. It is quite probable that some

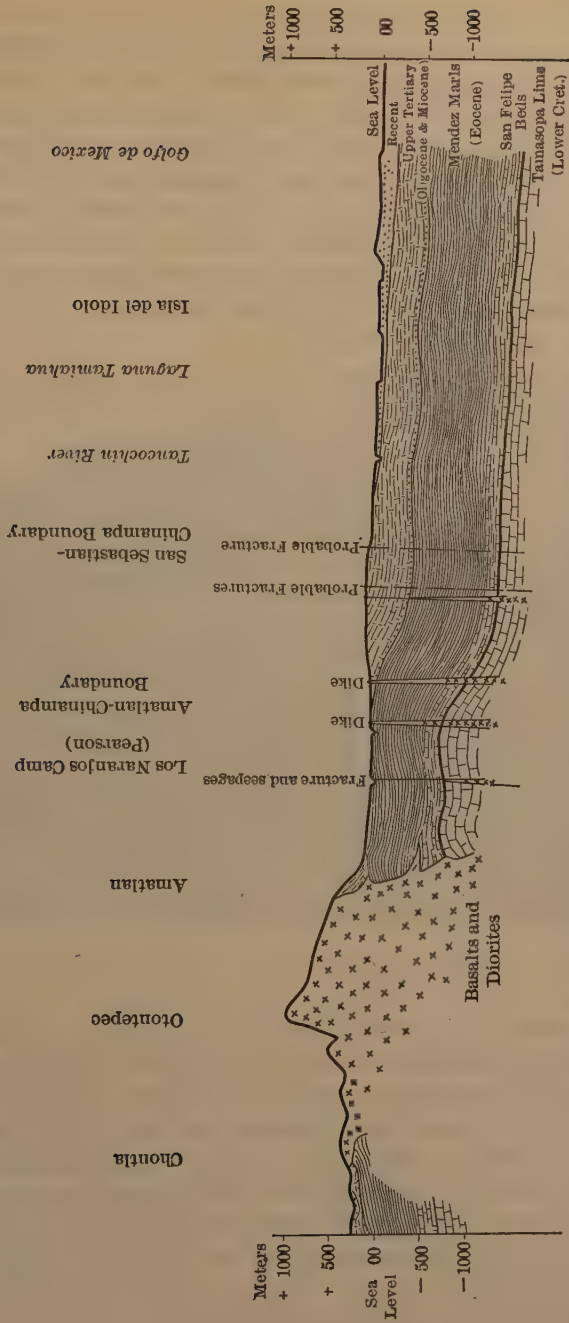


FIG. 7.—DIAGRAMMATIC GEOLOGICAL SECTION ACROSS LOWER PART OF FIG. 6.

such intersections, not filled with basalt to the surface, offered a channel for the circulation of underground water more or less hot or highly mineralized, which metamorphosed the sedimentary formations in the immediate vicinity. These formations would thus resist erosion much more readily than the softer limes and sands about the channel. One notable example of this is Cerro de Zaragosa, between Amatlan and Zacamixtle. This has every appearance of being a typical basalt peak, yet examination failed to show any basalt on its sides, which are composed of upper Tertiary formations. And yet the peak is directly in line with a main series of dikes extending from near Dos Bocas to Zacamixtle, through Juan Casiano and Los Naranjos. (Fig. 6.)



FIG. 8.—CERRO TANINUL, A CONICAL BASALT PEAK TYPICAL OF THOSE OCCURRING AT MANY POINTS IN THE MEXICAN OIL FIELD.

Another example is the eastern peak of the two on the edge of Aguada, called Taninul. The western one is composed of basalt (Fig. 8), while the smaller eastern peak shows only large blocks of a metamorphosed marl containing numerous small siliceous nodules. This peculiar rock is found at a number of places in the vicinity of large intrusions of basalt, and has been caused by the circulation of underground currents, probably both hot and more or less mineralized, as pyrite is frequently found nearby. There are also numerous smaller peaks and conical knolls throughout the field.

3. *Source of the Oil*

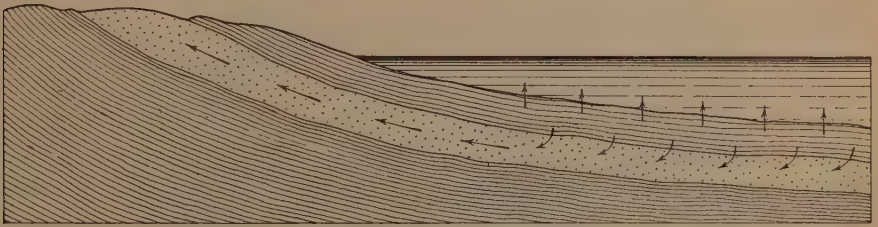
In a consideration of the source of the oil found in the wells in the Mexican fields, the following facts hold true at most points:

1. The oil is found in a porous and usually fractured limestone (sometimes shale) near the top of the Tamasopa limestone formation. Although a few shows, and, in some cases, large quantities of salt water, have been encountered, no oil in large quantities has ever been found as yet by drilling deeper into the lime.

2. Oil is also found, particularly in the "gusher" wells, in the broken lime shells and blue shales of the San Felipe series, usually under conditions indicating strong fracturing and jointing.

3. Oil is not found in quantity in the homogeneous marls overlying the San Felipe, although these marls are more or less petroliferous throughout, as shown by drilling. However, in drilling near dikes and fractures when seepages occur at the surface, shows of gas and heavy oil are often encountered in the hole all the way down.

4. These marls were laid down upon already compacted limestone beds of great thickness (Tamasopa) in which, so far as now proved, no



After King, 19th Annual Report, U. S. Geological Survey, part 2, p. 80.

FIG. 9.—DIAGRAMMATIC SECTION SHOWING THE FLOW OF CONNATE WATER, OIL, AND GAS, DUE TO CONSOLIDATION OF SEDIMENTS.

porous horizon existed. It is not considered probable that oil could have been formed within massive beds of marine limestone such as these.

5. Any oil coming from below the limestone must have been liberated by fracturing and intrusion and subsequently caught and held by the denser marls, pinching together and retaining such oil in the more easily shattered and hence more porous San Felipe and upper Tamasopa lime. Circulation of underground waters along the unconformity on top of the Tamasopa may have aided in forming reservoirs for the oil. There is considerable evidence of these channels in some wells, but they may have been formed after fracturing had promoted circulation.

This (5) presupposes either the inorganic origin of oil, or the existence of enormous previous accumulations of it in formations at great depth—below the Tamasopa. The latter alternative does not seem reasonable, because such previous accumulations would already have been more or less localized in certain areas, and when allowed to filter upwards through say 10,000 ft. of limestone after later fracturing, would have shown much more varied composition, and would have been restricted to more local

and erratic areas in the upper formations in the Mexican fields, instead of being so generally distributed under so nearly the same conditions.

The author holds it much more reasonable to assume that the oil had its origin in the petroliferous marine marls of the Mendez formation. The compacting of these marls under a constantly increasing overburden, as demonstrated by King (Fig. 9), would have forced the liquid content into any more porous bed existing below. It must be remembered that such marls are not impervious to the movement of fluids at the time of compacting, although they gradually become relatively so. As there is evidence of much channelling by hot water along the top of the Tamasopa, these channels offered both a course for the necessary movement, and the porosity for concentrating such oil as found its way into them. As stated by Roswell H. Johnson, "If beds of less compressible material (porous) meet or underlie the shales in question, there will frequently be a lateral movement along such beds to some point where upward movement will be resumed." There may be even today such compacting and lateral movement still going on along the shores of the Gulf of Mexico, which would be most actively manifested along fracture and fault lines where circulation upward could be resumed most freely. The greater the opportunity for this movement, while at the same time there was a chance for the segregation and retention of the oil, the larger and more extensive would be the final accumulations. Later movement along bedding planes of the marls probably found ideal catchment areas in fractures which had been formed in the meantime, and added to the accumulation of oil already there. This may account for the different composition of oils in the different districts.

Many phenomena noticed in connection with the production of oil in these fields are worthy of study. Employees of the Mexican Eagle Petroleum Co. report that when sudden barometric changes occur, such as those accompanying the frequent "northers" on the Gulf coast, the closed pressure of the Potrero del Llano well No. 4 increases as much as 100 lb., while nearby wells also show an increased pressure of smaller extent. If true, whether this indicates a connection between the underground reservoir and the Gulf through channels or fractures in the hard lower beds, or (what is more likely) direct connection of the oil and water table to the surface through fractures kept open by intrusives, is a question upon which more data are needed.

a. Basalt.—A reference to Fig. 2 will show that while the master intrusion in the central part of the Mexican fields is the Sierra Otontepec, and that radiating and tangential dikes in the immediate vicinity may be due to secondary fracturing caused by this large intrusion and its subsequent shrinkage; yet in general this represents only an exceptionally large example of a type common throughout the field.

So far as worked out by the author (Fig. 6) the long fracture lines

(usually filled by basalt dikes) were the primary phenomena. These are intersected at various angles by shorter fractures which resemble strong joint cracks at the surface, and which are less frequently filled by igneous intrusions. These are sometimes filled by sandstone dikes, as in the vicinity of Juan Casiano, and are in places indicated by short series of asphalt seepages following a general direction. As at such intersections there are so frequently found conical basalt peaks, or even hills of metamorphosed sediments, one soon comes to the conclusion that these knobs were merely phenomena incidental to and accompanying the main dikes. There is a group of such hills at La Pithaya and Juan Casiano.



FIG. 10.—OLIVINE-DOLERITE. MAGNIFICATION ABOUT 22 DIAMETERS.

Augite, olivine, and magnetite in well-formed crystals, and evidently crystallized before the feldspar. Labradorite in hemimorphic lath-shaped crystals surrounding the other constituents.

In neither case has the size of any one of these peaks, nor the distortion of the sedimentaries accompanying it, been enough to cause fractures of such length as are known to extend from these localities both north and south. The immediate presumption after studying these localities is that such fractures are the main fact to be taken into consideration.

Now, it will be noticed in Fig. 6 that Otontepec lies at the intersection of the two main series of dikes found in the central district—one extending approximately 20° W. of N. toward Panuco, and the other at about the same angle east of north toward Dos Bocas and the Gulf. These fractures are so persistent that the same conclusion appears to follow as was stated in the preceding paragraph, viz., that Otontepec is merely



FIG. 11.—SPECIMEN FROM A SMALL DIKE. MAGNIFICATION 40 DIAMETERS.
Larger proportion of augite than Fig. 10 or 12.



FIG. 12.—OLIVINE-DOLERITE. MAGNIFICATION 22 DIAMETERS. SAME AS FIG. 10.



FIG. 13.—BASALT DIKE APPEARING AT THE SURFACE IN THE MEXICAN FIELDS, NEAR OZULUAMA.



FIG. 14.—PORTION OF A LARGE SEEPAGE OF "CHAPOPOTE" IN THE CENTRAL DISTRICT OF THE MEXICAN OIL FIELDS.

Notice the remains of the cow that became entrapped in the bed of asphalt.

an intrusion which represented the point of greatest weakness in the sedimentary rocks. It probably was formed by the coalescing of a number of necks which found their way to the surface as did the smaller ones found at other points. A study of the districts south and west of Oton-tepec would probably yield confirmatory evidence. As it is, there are no published data on these districts.

The accompanying sections (Figs. 10, 11, 12) were taken from specimens of basalt dikes in this central district, by W. H. Tomlinson.

b. Seepages.—Seepages of asphalt, sometimes accompanied by gas and occasionally by sulphurous brine, are found throughout the Mexican field, and, with the relatively horizontal strata and igneous intrusions, constitute a unique feature of this oil field. Several of the largest groups of seepages (Fig. 6) are found on anticlines, and most of them are near dikes or intrusions which reach the surface. Others, however, appear to exude from the surface formations where there is apparently no structural reason for oil accumulation below. However, experience and inference have led to the general belief on the part of geologists that in most instances seepages indicate an intrusion of igneous rock in the vicinity not far from the surface (Fig. 14).

It is common to find short lines of seepages intersecting a main dike at a high angle (Fig. 6). A series of such short lines of seepages, each at an angle to the strike of that district, yet taken together following the strike of a dike, is often evidence by which a fracture can be projected from another locality. The shorter intersecting fractures probably in most cases represent joint cracks due to local pressure. The photograph (Fig. 13) shows a spur of basalt from the dike following such a direction.

Seepages are also found exuding from basalt high on the Sierra Oton-tepec, at a considerable distance from the contact of the sedimentaries. As there are many open channels through the basalt which would obviously offer less resistance to the passage of oil than would the overlying Mendez marls, these apparently anomalous occurrences are not so in reality. Basalt dikes cannot be looked upon as barriers to the movement of oil, except so far as they accompany fractures which tend to entrap it. Such migration may be from several directions.

A study of the central district shows a significant resemblance between the fractures located and a series of joint or shrinkage cracks which might be due to a compacting and drying of the sedimentaries. A remarkable persistence of roughly triangular areas inclosed by these fractures, and the fact that their persistence in certain general directions has some relation to the direction of the main stresses at the time of the Sierra Madre uplift, suggest that such general jointing or shrinkage cracks may have been the primary cause of the lines of weakness which were followed later by intrusions.

4. *Limits to Prospective Fields*

In the area between Las Palmas and Micos, some sharp anticlinal structures occur, in one of which several wells have been drilled (San Pedro), with a small production of light oil (45° B \acute{e} .). This, no doubt, will be further prospected; but as the production per well will probably continue small, and as operating is very expensive, it will probably not pay to go after such risky prospects when the low-gravity oils are so plentiful. The cost of producing this higher-grade oil will make it more profitable to import such refined products as cannot be made from the lower-gravity oils of the eastern fields.

The Topila district will probably be developed into an area where the salt-water problem will not be so serious; and probably such areas as Caracol and Los Esteros will be finally developed into better producers, although the first wells have been disappointing. There is then a great area north of Tampico, along the coastal plain between the Panuco River and Texas, in the part of which south of Soto la Marina (San Jose de las Rusias), at any rate, prevailing conditions are similar to those in the southern region. The Dutch Shell interests have large holdings in this northern area. There are many seepages and igneous intrusions in a part of this region.

The district between Las Palmas and Ebano and thence south to Tantoyuca, has not been prospected, owing largely to transportation difficulties. Before the present political difficulties, a railway had been begun, to connect by a short route between Tampico and the city of Mexico. When completed, this will open up the area mentioned. While in general this area does not seem to be so favorable as that east of Otontepec, yet not so much is known of its geology and structure, and it is probable that good pools will be developed, possibly of lighter oil than that of the present fields.

In the extreme eastern belt, along the coast between Tampico and Tuxpam, owing to the fact that the full thickness of the upper Tertiary formations must be passed through, together with considerable thickness of still more recent deposits, drilling at many points will have to be very deep before encountering either the San Felipe beds or the Tamasopa lime. At Tanguijo the upper Tertiary beds are at least 1,200 ft. thick, and a well 4,000 ft. deep had not passed the San Felipe. Moreover, the frequent deficiency of surface indications through the relative scarcity of seepages; the lack of intrusives appearing at the surface; and the possibility of seepages having travelled a considerable distance through the extra thickness of upper formations before appearing at the surface;—are all factors making it difficult to locate wells with the same chance of success as elsewhere. Seepages here are not so significant of accumulations or strongly fractured zones directly below, as in regions where the upper

beds are thinner. It is also argued that so far down the dip to the eastward there is more danger of encountering salt water. This is not so dangerous as would at first appear, for oil would be caught in the upper part of fractures, while the water would encroach along the lower portion, and thus "spill" around the entrapped oil, much as happens in the case of encroaching water passing and retaining the oil in the crest of a sharp anticline. Such wells might possibly be shorter lived.

V. PROBABLE FUTURE OF THE MEXICAN FIELDS

1. *Evidence of Producing Wells*

The first important producing well in the Mexican fields was the Mexican Petroleum Co.'s Pez No. 1 in the Ebano district. This well has been producing since 1904, and still continues to flow with little diminution at more than 1,000 bbl. per day.

The famous Juan Casiano No. 7 has been producing continuously since September, 1910, and is now making 700,000 bbl. per month, and has approximately 40,000,000 bbl. to its credit.

Nor are these isolated cases. The Pearson No. 4 well at Potrero del Llano has done at least as well as Casiano No. 7 for about the same length of time, and the older wells at Panuco are still good producers in the most closely drilled field of all. While the Chila-Salinas and Topila fields have suffered from the invasion of salt water in the past, yet recent development work indicates that adjoining areas may be better in this respect. The Dos Bocas gusher blew a great crater in the ground, and after producing 200,000 bbl. of oil per day for 57 days, went to salt water. Large quantities of asphalt are still thrown out with the salt water and gas from this hole, and there is reason to believe that if this well had not got beyond control it would have held back the salt water until its oil production would have been much greater in the aggregate than the actual amount expelled during its short life. As already observed, the larger wells in a number of districts are inclined to increase their production after being allowed to flow wide open for a short time. All the evidence of past wells points to the long life and high productivity of most of the districts already tested, and the probability of many more being eventually opened up.

2. *General Summary*

It is believed that Mexico is potentially the second greatest oil-producing country in the world today. The reasons for this belief are summarized below, as well as the causes which will tend to hold back for some time the realization of the possible maximum production:

1. A large number of widely separated areas have been tested and have developed uniformly large wells.
2. With the exception of a few initial wells, none of these areas has been fully developed by the drill. There is thus a considerable acreage undeveloped in already proven territory.
3. Numerous promising localities remain untested in the present fields.
4. Large districts yet untested, such as those north of Tampico and west and northwest of Otontepec, offer surface evidence of the same characteristics as the proven fields.
5. The production per well has been large, and the decline slow in most fields.
6. The underground accumulations in fractured zones are capable of yielding a much larger percentage of their petroleum content than is a sand body.
7. More efficient methods of production result from the usual control of each producing district by one large company.

If such districts were located in the United States, they would soon be drilled up to a production such that the price would break to a very low point; but in Mexico this maximum production will doubtless be reached much more slowly, because of: (1) internal political difficulties; (2) delay in opening up isolated districts by railroads and pipe lines; (3) the many large tracts of land in most districts, the expense of handling which keeps out small operators, who would over-drill; (4) the high expense of operating, which keeps out a host of adventurers; (5) the lack of tankers for transportation; (6) the difficulty, by reason of expensive production and transportation, of competing in the United States market with Gulf Coast and California fuel oils; and (7) the present over-production of high-grade oil in the United States, which will delay the adaptation to Mexican oils of the new processes for refining heavy oils. These economic conditions account for the fact that Mexico, from wells having a present daily capacity of from 330,000 to 500,000 bbl., is producing only 70,000 bbl. a day, and that no active endeavors adequate to this discrepancy are now in progress, for the liberation and conveyance to the sea-coast, of the enormous amount of oil thus shut in.

Sliding Royalties for Oil and Gas Wells

BY ROSWELL H. JOHNSON,* PITTSBURGH, PA.

(San Francisco Meeting, September, 1915)

THE principle of sliding or graduation in royalties is accomplished either by the block, period, cumulative, or class method.

The block method calls for a very low royalty rate on all oil produced up to a certain amount. This amount would normally be that which is believed to suffice to repay the cost of the well. A higher rate would be charged after the well is making a profit. This method, common in many industries, is peculiarly inadaptable to any industry such as this where the productivity per day of the units gradually and inevitably decreases. This, for reasons given later, is because it forces an early abandonment of the well and so offends against conservation.

In the period method, the royalty rate is changed by some definite amount when the well produces less than a specified quantity per day or other given unit of time. While simple to explain, it operates badly because a great deal is made to depend upon a small reduction in production, whereas the reduction as a matter of fact declines rather erratically owing to the exigency of lease management, connecting of tankage, and vagaries of the gaugers.

These disadvantages are obviated by the cumulative method. In this method, all the production less than a certain amount per week has one rate of royalty, or none at all, and the oil produced in excess of that amount pays an additional rate of royalty. In the long run the calculation is probably no greater than with the period method, for simple tables will be published and printed in the oil papers that can be cut out, enabling quick readings to be made. Even without the table the calculation is easy with the few royalty rates, two or three, which would be used. In this method, the amount paid in royalty declines gradually, rather than suddenly as in the period method.

The following reasons might be advanced for the adoption of cumulative royalties:

1. To distribute the burden in accordance with the capacity to pay. This principle is a unique one in business, although familiar in taxation. This is because it does not distribute reward in proportion to efficiency.

*Johnson & Huntley, Consulting Geologists.

To fail in this is to allow an industry to degenerate in its progressive and alert attack on the technical and economic problems of the business. It is not, then, to produce a more equalitarian and less just distribution of wealth among producers themselves that the cumulative royalty is here urged.

2. The fear has been expressed that if sliding royalties are adopted, too steep a graduation might be used, which would have the effect of destroying the occasional high rewards essential to the producer to recoup him for the heavy expenses of the inevitable proportion of dry holes. Such a result would necessitate a great increase in the cost of oil to the consumer to produce higher profits on small wells in order to cause the producer to continue his activities. Such a result would be a social loss, but the fear in my opinion is not justified, for the land owner would not get such steeply sliding royalties without serious sacrifice in the bonus, which he would usually prefer not to make.

3. Another advantage of the cumulative royalty is to prevent the excessive flat royalties now frequently offered for promising land, such as the 50 per cent. on the Cimarron River bottoms and the 25 per cent. that we occasionally hear of in other fields. These royalties always lead in a few years to an unpleasant threatening and bargaining between land owner and lessee resulting in successive agreements to reduce the royalty rate. This awkward process, it is true, does accomplish a gradual reduction of the royalty rate. But the asperities of such negotiations are very annoying and sometimes lead to a premature abandonment of the well, with a consequent serious offense against wise conservation. The cumulative royalty, while avoiding the difficulties just referred to, still retains a flexibility by virtue of which the producer who desires to may increase the royalty rate, when the well can stand it, in this simple way transferring some of the speculative profits to the land owner in lieu of bonus. This bonus the producer may not be able to pay, or the land owner may prefer not to accept.

4. The fourth reason is the most weighty. The cumulative royalty lengthens the life of the well and increases the percentage of the oil which is recovered. If the royalty is one-sixth and the maintenance and interest on the "junk" is $83\frac{1}{2}\text{c.}$ per day, then a well must be abandoned when its net income to the producer reaches that amount. Yet the gross income is still \$1 per day, and if the decline of the well is one-sixth in a year (a common decline in old wells in Oklahoma) it might continue to produce for a year longer except for the royalty. Thus, 300 bbl. per well might easily be saved by a mere royalty adjustment. If the Osage Nation is leased at one-sixth, as now threatens, all its wells will be prematurely abandoned and a most serious loss result.

A fourth method using a sliding rate is the class method. Here the wells are classified at the beginning and a different rate used with each

class. The classification is based upon the ratio of value of product to the cost of production. Normally some one variable factor in either of the items above would be the basis of the sliding. The one which will be most used and which I recommend is the depth of the well.

To charge the same royalty rate for wells of 600 ft. as for those of 3,000 ft. has the effect of promoting "post-hole" drilling, as we call holes of inadequate depth. Such a wide distribution of shallow dry holes, where the untouched underlying strata are worthy of test, results not only in a direct waste, because the area must later be redrilled, but also in an indirect waste, because later operators, being in doubt as to the depth of the older holes, are afraid to risk drilling in territory thus improperly condemned. The classes should be few, and all wells in one pool should be in one or another of the classes.

For instance, if one-sixth royalty be charged in the Osage Nation then the deeper sands of the western Osage would not be as systematically or economically prospected, as would be the case if the royalty was graded by depth. I suggest one-sixth for oil in pools averaging above 2,000 ft.; one-eighth from 2,000 to 3,000 and one-twelfth below 3,000, for the present leases with their partly developed production. For the rest of the Osage Nation I would suggest one-eighth above 2,800 ft. and one-twelfth below it.

Let it be remembered that the Mississippi lime (Boone chert) lies deeper than 3,000 ft. in much of the western Osage and that no test is thorough unless the drilling is continued until that formation is reached, when the upper horizons prove barren. Let no one suppose that because the Cushing and Boston pools have been great producers all the deep pools of the Osage will be of that kind. West Virginia has plenty of deep small producers, and so has the Bartlesville sand. The system above outlined would encourage thorough prospecting.

The objection that one-twelfth and one-sixth are too great a difference will arise in the minds of those who think only of successful wells. It seems none too great when one remembers the amount of futile drilling which must be paid for by proceeds from those that are successful. To consider next the requirements of a proper system of sliding royalties:

1. It must be simple, so that it can be readily understood.
2. It must not be expensive or difficult to calculate.

It follows from these two considerations that it should have but few rates.

3. It must avoid uncertainties as to the time or point of changing rates.

4. Most important, it should permit the well to be pumped until its gross income has fallen to its maintenance cost.

To accomplish this last, I suggest an exemption from royalty on the oil equal to maintenance and interest on the junk. The slight loss to the

land owner will be met ordinarily in the bonus, but sometimes by a higher royalty. For practical reasons the amount that should be exempted, instead of being exactly equal to maintenance and interest on the junk, would be fixed for the field at an approximate integral number of barrels, ordinarily one. When several wells discharge into one field tank, they may be averaged.

5. It should not encourage post-hole drilling as opposed to thorough prospecting.

6. It should bear some relation to the cost of production.

For these two reasons we should in addition classify the royalty according to depth.

The objection which might be raised that the land owner would in some cases receive nothing for the use of the land during the last months of the history of the well is met by the fact that he has received advance payment for such use either in bonus or in large royalty during the early history of the well. It would be absurd to say that it would be illegal to pay a lecturer a royalty only on the tickets sold after other costs have been met.

Gas Royalties

In some fields gas may be so cheap and wells so isolated that it is still infeasible to meter from the lease. But where a gas royalty is paid or indicated as the better method, the general principles laid down above apply in even a greater degree than to oil.

This is because a gas well is so frequently abandoned because its volume is not enough to pay the fixed rental for a well; also because it is the small wells about to be abandoned which have especially heavy maintenance by reason of the necessity sometimes of pumping water, or more often of installing pumping stations.

To meet this situation I recommend for gas wells a stated royalty, with the exemption of a certain amount per week, roughly estimated in advance, equivalent to cost of maintenance and interest on junk. This exemption is to be increased to a higher proportionate exemption if, and when, water pumps are installed at the well or pumping plants are required to raise the pressure sufficiently to put the product into the main lines.

We conclude then that the flat royalty offends against a wise and economic exploitation of our oil and gas deposits.

DISCUSSION

WILLIAM A. WILLIAMS, San Francisco, Cal.—A sliding scale of royalties was proposed to the Department of the Interior in connection with the fixing of royalties to be charged upon oil leases made on the Osage Reservation in Oklahoma. The possibility of evasion and difficulty of

enforcement required close inspection to satisfy all interests, particularly in connection with the exemption from royalty on oil to equal maintenance and interest on the junk.

Mr. Johnson discusses the block method, the period method, and the cumulative method, and favors the adoption of the cumulative method with a proviso that the wells should be permitted to be pumped until the gross income is as low as the maintenance cost, which would mean exemption from royalty on the smallest wells. He recommends that depth be the variable factor upon which the sliding royalty is based and suggests that on partially developed territory in the Osage, the royalty be one-sixth for oil wells averaging less than 2,000 ft. in depth, one-eighth for wells from 2,000 to 3,000 ft., and one-twelfth above 3,000 ft.; for the rest of the Osage he suggests one-eighth above 2,800 ft. and one-twelfth below that depth. The net result, if Mr. Johnson's suggestions had been accepted in the Osage, would have been to reduce the then existing rate of royalty and considerably increase the cost of administration.

The difficulties in determining the size of wells to be exempt and the possibility of obtaining oil at different depths in the same well might lead to complications which are avoided by charging a flat royalty as now universally charged throughout the United States. It is not uncommon to reduce the royalty when the production of the wells becomes unprofitable at an existing rate of royalty, accomplishing very much the same results as Mr. Johnson has attempted to accomplish by the cumulative royalties, and probably in a manner far more satisfactory to the operator and the lessor.

A. C. McLAUGHLIN, San Francisco, Cal.—The net revenue of the well depends, first, on the cost of production; second, on the amount of oil produced; therefore it would be impossible under Mr. Johnson's system to set a royalty which would be fair to all concerned. If it is impossible to set it at the beginning, it is certainly further impossible to do it from day to day. I think it is entirely impracticable of application.

WILLIAM A. WILLIAMS.—Mr. Johnson said this would be based on pipe-line statements of the runs.

A. C. McLAUGHLIN.—But the price obtained for the oil and the cost of production are big factors. I know of some wells with an average production of 150 bbl. a day, and costing to operate say, \$150 a month, each; but when they get to be strippers, the operating costs are \$25 a month. It is impossible to set an equitable rate of royalty on that basis.

A. F. L. BELL, San Francisco, Cal.—It seems to me that on Government lands in California a sliding royalty should be a dangerous thing

to inaugurate in the oil business. I think there should be a period royalty.

I made an examination of about 500 wells, in one of the Oklahoma fields, and found the average life to be about 7 years, after which the wells produced less than a barrel per day.

In California the production of the well would be greater and the profitable life larger. Whatever we do let us try and keep away from a sliding royalty since it would add to the already heavy burden of the oil companies.

ROSWELL H. JOHNSON (communication to the Secretary*).—The paper consists of (a) an analysis of the different methods of graduating royalty, (b) an examination of each as applied to oil wells, (c) two positive recommendations.

Has not Mr. Bell confused the complexity of the types of sliding (which I mentioned only to reject for that very reason) with the two kinds I favored?

I have recommended that a different rate be used in fields where costs are light and receipts heavy than that used where costs are heavy and receipts light. This is merely to yield to what is already a tendency, and what is very general in nearly all businesses that have royalties; the royalties of authors, lecturers, playwrights, mine owners and others vary according to just such considerations of costs and receipts. It is more scientific to vary the royalty than to have a universal fixed royalty and varying bonuses, because lower royalties in expensive regions are necessary to make developments in such regions possible.

My other recommendation is that a definite number of barrels (in some fields it would be one) be exempted from all royalty payment in order that the life of the well may be continued. This seems to me to be extremely simple as compared to many sliding arrangements that are in widespread use. In fact, it is one of the simplest and easiest that can be devised, and gradually lowers the royalty rate step by step as the well declines. It is, technically, a truly "degressive" rate.

Mr. McLaughlin very properly contends that I should have specified the time in the history of the well at which the cost of maintenance, etc., is determined. That time is when the well is old and nearing abandonment, since the main purpose is to prolong the life of the well. The fact that maintenance is higher per well earlier need not concern us.

The contention that late maintenance is variable is unimportant also, since we cannot hope entirely to prevent unduly expensive wells from being abandoned earlier. The amount should be fixed at the nearest even number of barrels above the cost of maintenance and "interest on

junk" of the *average* old wells in that district. To be sure, price per barrel changes, but when it is higher, costs are in general higher, so that it is in some degree compensatory.

In regard to the Osage wells, the fact that at times some wells operate at less than capacity need not interfere in any way, since there is all the more reason to try to assist in keeping small wells alive during such times of stress, as it is at such times that we get most of the premature abandonments of wells that are needed later.

I see no difficulty in the way of enforcement, for we simply subtract a constant number from the pipe-line runs before computing the royalty. The well need not be seen.

There is on the other hand an administrative feature in the classification of well by depth (or other element in the cost of receipts). However, the depths of the several sands are so well known that in very few cases would the operator dare to return a false statement, even should he be inclined to do so.

The government, in insisting on one-sixth royalty in the Osage, no matter how small, old, or deep the wells on a property are, seriously offends against conservation. It should be urged to consent to a reduction of royalty in the leases now existing when their wells are near abandonment, and to avoid this unfortunate situation in the new leases soon to be bid for by making express exemption from all royalty of a barrel a day. The Indians would be compensated by the higher bonuses that would be bid.

Oil, Gas, and Water Content of Dakota Sand in Canada and United States

BY L. G. HUNTLEY,* PITTSBURGH, PA.

(San Francisco Meeting, September, 1915)

Introduction

IN view of the recent advance made in the knowledge of the nature and conditions accompanying the occurrence of oil and gas, and of the recent activity in drilling in Wyoming, Montana, and western Canada, a discussion of the probabilities of obtaining production in such a reservoir as the Dakota sand in general, and in particular districts, is thought by the author to be timely and likely to result in more intelligent operating in certain parts of the territory underlain by this sand. In Canada particularly it is thought that these general considerations have equal if not more significance in a consideration of the oil and gas possibilities of the formations than have the various local conditions.

Age and Nature of Deposition of the Dakota Sand

The Dakota is distinctive in that it is known to form a generally continuous sheet of sand underlying the greater part of Alberta, Saskatchewan, Manitoba, North and South Dakota, Montana, and large areas in Nebraska, Colorado, and Wyoming. It does not seem to have been laid down in more or less isolated lenses, as has been the case with most oil-sand horizons.

Except in northern Alberta, the Dakota sand contains fresh-water fauna, having been deposited very uniformly over these wide areas in shallow water with prevailing strong currents. In northern Alberta marine and brackish water conditions prevailed. In this district it forms the basal member of the Cretaceous sediments, directly overlying beds of Devonian limestone along a great unconformity in the Athabasca River valley and eastward. In southern and western Alberta it is underlain unconformably by the Kootenay shales (Lower Cretaceous), and by Jurassic and Carboniferous rocks. In the United States in South Dakota it was in places deposited upon Cambrian quartzites, and in other places directly upon the granite. In the State of Dakota, and in the

* Johnson & Huntley, Consulting Geologists.

outcrops about the Black Hills uplift, the Dakota sand is underlain by from 200 to 350 ft. of sandy shales comprising the Fuson formation, the Lakota sand, and the Morrison shales, all of Lower Cretaceous age.

Westward from the Black Hills these formations thin out, and in the Big Horn Mountains are represented by the "Cloverly" formation. It is believed the "Cloverly" includes the Dakota, the Fuson formation, and the Lakota, lying conformably on the Morrison shales (Jurassic or Lower Cretaceous).

The principal point to be brought out is that in most of this area, except as mentioned above, the Dakota was laid over an unconformity at the base of the Upper Cretaceous measures.

In the Athabasca River country of northern Alberta, the Dakota is overlain by the black carbonaceous Clearwater shales (Benton?), and by the Grand Rapids sands (Niobrara?). Some authorities include these in the Dakota group, while others consider them to be of Niobrara age. Overlying the Grand Rapids sandstone series are about 900 ft. of black carbonaceous LaBiche shales. In southern and western Alberta and Montana the Dakota is overlain by the Benton, Niobrara, and the Claggett or Lower Dark shales. These are all of marine origin, and are unbroken by any continuous sand horizons except a few irregular lenses of limited lateral extent. In southern Montana and Wyoming the Dakota is overlain by shales of the same general character, but included in them are a number of locally continuous sand bodies, some of which have proved productive of oil and gas. Among these are the Shannon sand of the Pierre, and the Wall Creek sand of the Benton, in the Salt Creek and Powder River pools in Wyoming; and the sandy Aspen formation in the lower part of the Benton shales, which contains most of the oil in the Spring Valley district in Uinta County, Wyoming.

Lithological Character of the Dakota

A typical section of the Dakota sand consists of a lower basal member of conglomerate followed by beds of coarse porous brown sand with some clay. The upper portion of the horizon is often split into lenticular sand bodies alternating with shale, and frequently indicates a return to brackish and marine conditions. In this upper portion are frequently found locally continuous beds of limestone. Only along the western shore line of the Dakota horizon at its outcrop in the foot hills and in the Moose Mountain district in western Alberta and British Columbia, is there a pronounced change from the general section given. Here the basal bed of conglomerate and coarse sand persists, but this is overlain by shales and quartzitic sandstones to a thickness of almost 1,000 ft.

This remarkable uniformity of outcrop indicates a continuity which is further attested by many wells drilled throughout this area, as well as

by the generally artesian character of the Dakota sand throughout North and South Dakota and central Canada. Its character places it in the category of what Roswell H. Johnson describes as a "sheet" sand, including such sands as the æolian St. Peter's sandstone, and the Sylvania sand of Ohio. That is to say, it is an actually continuous sheet of porous sand with no extensive shale bodies interrupting its continuity, as opposed to a horizon such as the Bartlesville sand in Oklahoma, which is a name given to a lateral series of more or less connected sand bodies or lenses, occurring at a certain position in the columnar section.

As stated previously, the economic evidence of this sheet nature of the Dakota sand is its having the general character of an artesian sand reservoir underlying the great synclinal basin west of its outcrop in Dakota and Manitoba. As such, it is one of the valuable natural resources of these districts.

The Dakota thins out in central and western Wyoming, and shows considerable cross-bedding along its outcrop in the Big Horn Mountains. In parts of this area it consists of a series of thin sandstones alternating with beds of shale. On this western edge it is considered as being included in the Cloverly group of sandstones, which probably includes the Fuson and Lakota formations of Lower Cretaceous age. (G. H. Eldridge: *Bulletin No. 119, U. S. Geological Survey*, p. 22.)

Extent of the Dakota Outcrop in Canada

The outcrop of the Dakota in the Moose Mountain district has already been mentioned. Besides the outcrops along the Athabasca River, few exposures have been noticed in northern Saskatchewan and Manitoba. (Wyatt Malcolm: *Memoir 29-E, Geological Survey of Canada*.)

1. On Carrot River, 4 miles above the west line of Indian Reserve.
2. Along the south shore of Lake Wapawekka, to the east of Lac LaRonge.
3. On Beaver River, just above the mouth of Dore River, banks of white sand 90 feet high.
4. South shore of Isle de la Crosse.
5. In various water wells drilled back of outcrop in southern and eastern Manitoba.

By referring to Fig. 1, it will be seen that from any point along this eastern and northern outcrop seepage conditions are down-dip to the great central Moosejaw synclorium. As the Dakota outcrops in a region of lakes and muskeg swamps southeast from the height of land, its being a reservoir filled with fresh water throughout wide areas is explained, drainage conditions having kept its exposure more or less saturated with entering surface water since Cretaceous time.

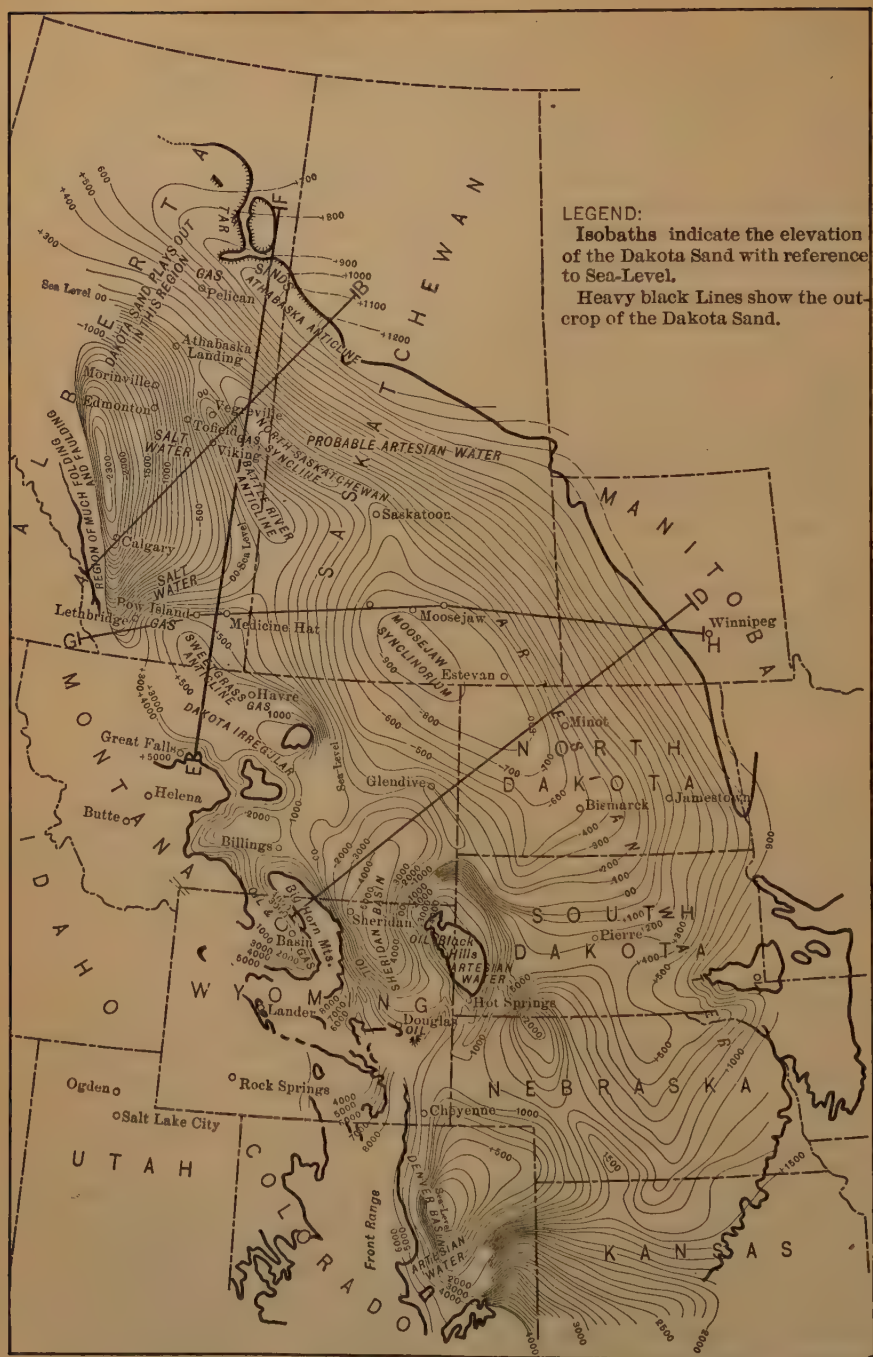
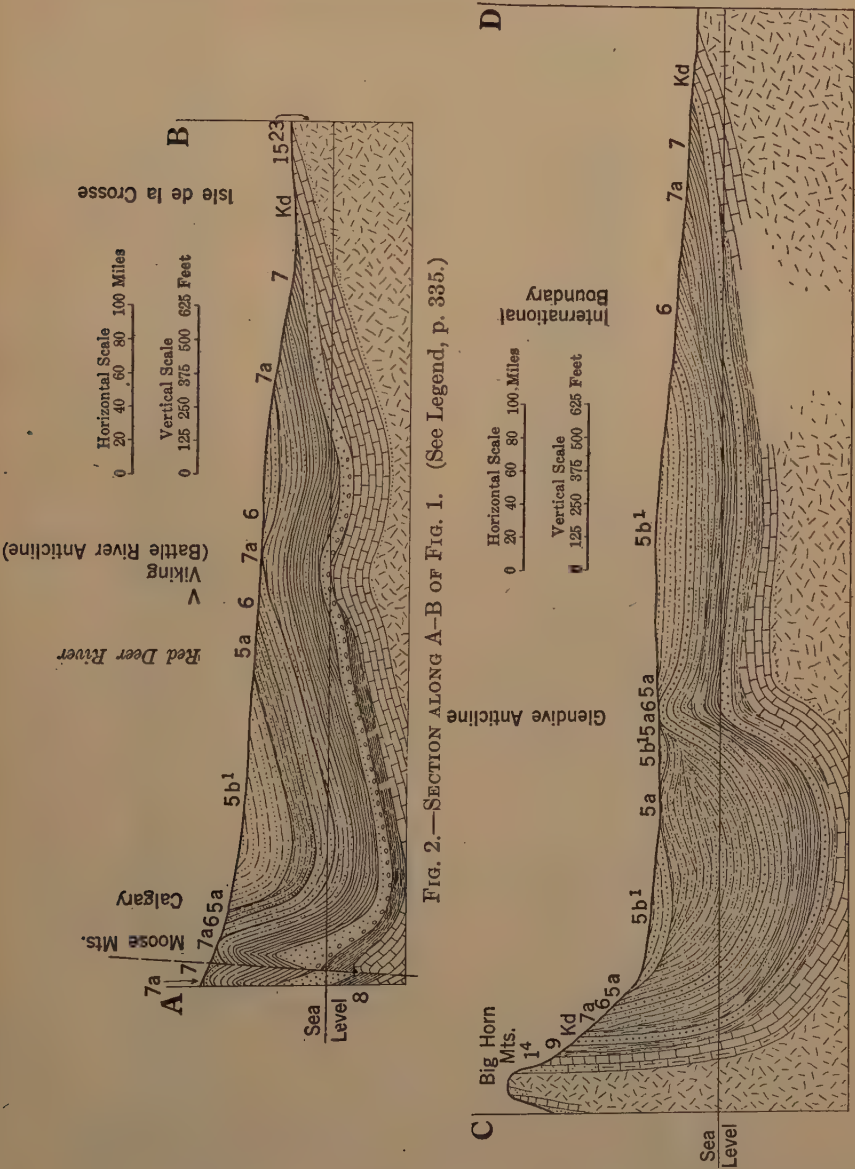


FIG. 1.—Sketch Map Showing Generalized Structure of the Dakota Sand in the United States and Canada, with Relation to the Oil, Gas, and Water Reservoirs in that Sand Body.

The Dakota sand plays out in the region between the Athabasca and the Peace Rivers. On the latter river it is apparently represented by the Loon River shales at the base of the Cretaceous measures. (Mc-



Connell: *Memoir, Canada Geological Survey of Canada*). It was also not recognized as a sand in the deep well drilled at Morinville, 18 miles north of Edmonton, Alberta.

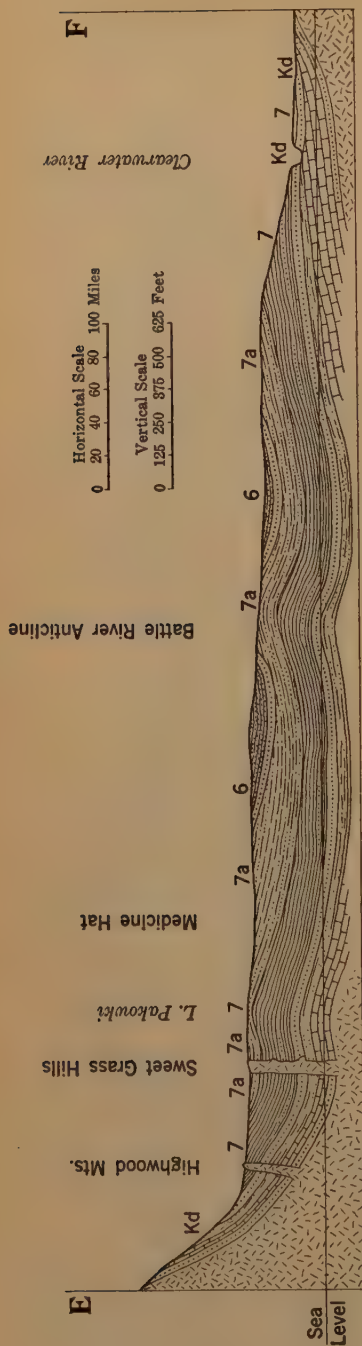


FIG. 4.—SECTION ALONG E-F OF FIG. 1.

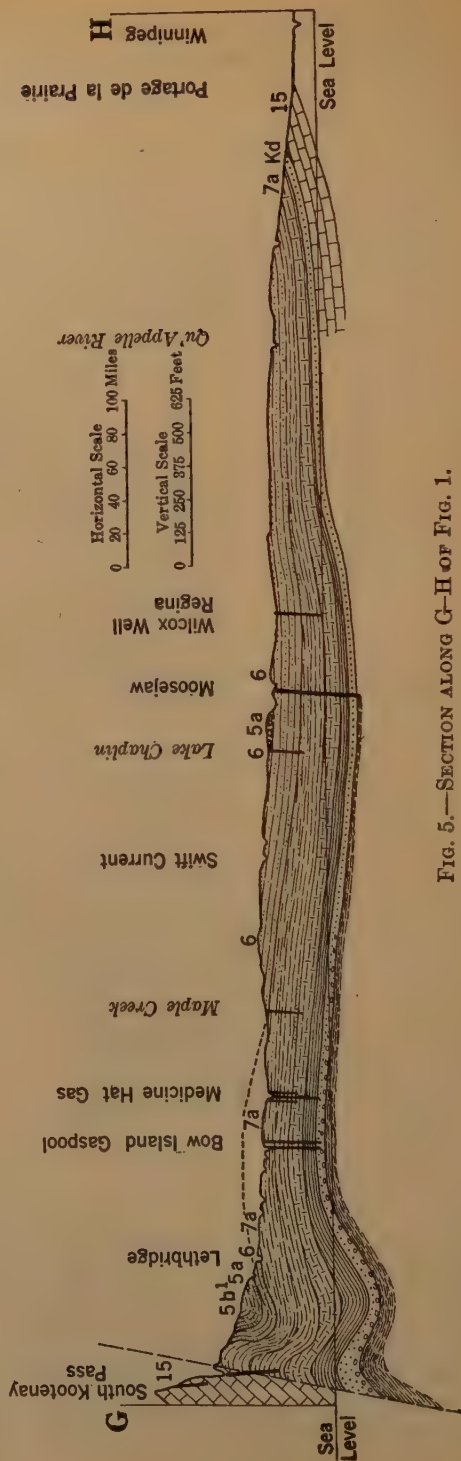


FIG. 5.—SECTION ALONG G-H OF FIG. 1.

Legend—(applies to sections, Figs. 2 to 5)

Lower Tertiary.	5b1, Laramie—Paskapoo Series.	Fresh-water sands, clays and shales.
Upper Cretaceous.	5a, Laramie—Edmonton Series (coal bearing).	Sand and shales.
Upper Cretaceous.	6, Bearpaw (Pierre-Foxhill).	Gray-brown shales, sand shells.
		Sand and shale in upper portion, black shale below.
Upper Cretaceous.	7a, { Belly River and Lower Dark shales.	
	{ Niobrara (cardiom).	Sand lenses and dark shales.
	{ Benton.	Black and gray shales
Upper Cretaceous.	Kd, Dakota sand.	Soft, porous sand (250 to 950 ft.), conglomeratic at base.
Lower Cretaceous.	8, Kootenay shales (coal-bearing).	
Devonian.	15, Devonian.	Limestones, shale and salt or gypsum.
Cambrian.	18, Cambrian.	Reddish sand and shales.
Archean.	23, Laurentian.	Granite.

Tar Sands, and Production of Oil from the Dakota Sand

Where the Dakota outcrops along the Athabasca River, from Crooked Rapids to Fort McKay, and in the tributary streams on either side, it is found to be saturated with inspissated petroleum or "tar," from its top to its bottom. This tar-saturated sand continues up the Clearwater River to the eastward, and it is also reported in the vicinity of Buffalo Lake across the line in Saskatchewan. A considerable "lake" or seepage of this "tar" is also reported in the country between the Athabasca and the Peace rivers just back from the probable outcrop (Fig. 1), and probably exudes from the Dakota sand a few feet below the surface. This seepage is reported to have been increasing in size in recent years.

A small amount of dry sulphurous "tar" is also found coming out of the old government well at Pelican Rapids along with the gas.

Petroleum has been produced from the Dakota horizon in the Powder River pool and the Big Horn Basin in Wyoming; and it is quite possible that deeper drilling in the Salt Creek pool will prove its existence at this horizon on that dome. Small seepages have been reported from the Dakota outcrop on the western slope of the Black Hills uplift, and a little heavy oil has been produced from the Moorcroft pool. (C. V. H. Barnett: *Bulletin* 581, *U. S. Geological Survey*.)

Analyses of Oil Produced from the Dakota Sand Horizon

The following analyses of oils produced from the Dakota horizon are given, as indicating what may be expected in different localities:

A sample of oil from the Moorcroft pool, Crook County, Wyoming, according to David T. Day, had a specific gravity higher than 0.9000. It is a heavy oil suitable for the manufacture of lubricants. Paraffin base.

Specific gravity.....	0.9198
Baumé, degrees.....	22.2
Color.....	dark green
Begins to boil.....	130° C.
Gasoline, per cent.....	0.5
Kerosene (0.858 specific gravity), per cent.....	24.0
Residuum (0.94 specific gravity), per cent.....	76.0
Paraffin wax, per cent.....	2.5

Average analysis of three samples of oil from the Powder River pool, analyzed by David T. Day:

Specific gravity.....	0.9123
Baumé, degrees.....	23.46
Begins to boil.....	213° C.
150° to 300° (0.8528 specific gravity).....	17 c.c.
Residuum (lubricating oil, paraffin and tar, 0.9295 specific gravity).....	81.2 c.c.
Total c.c.....	99.1
Water, per cent.....	0.9
Sulphur, per cent.....	0.395
Asphaltum, per cent.....	2.39
Unsaturated hydrocarbons:	
Crude, per cent.....	30.7
150° to 300° C., per cent.....	7.3

Analysis of oil from Well No. 1 of the Athabasca Oils, Ltd., 9 miles north of Fort McKay, Alberta. Analyzed by the Mines Branch of the Canadian Department of Mines, Ottawa:

	Per Cent.
Water, approximate.....	5
Gasoline and naphtha, 0° to 150°.....	10
Kerosene, etc., 150° to 300°.....	15
Residue.....	70

Analysis of oil from Well No. 2 of the Athabasca Oils, Ltd., 9 miles north of Fort McKay, Alberta. By Falkenburg & Laucks, Vancouver, B. C.:

Specific gravity.....	1.02
Gasoline and naphtha, 0° to 150°, per cent.....	7.1
Kerosene, etc., 150° to 300°, per cent.....	60.1
Base fixed at 300° C., asphaltum, per cent.....	32.8

Analysis of oil from the Wyoming Oil & Development Co. well on Sec. 8, T. 32, R. 73 W., in the Douglas field, Converse County, Wyoming. Made in the U. S. Geological Survey laboratory under the direction of Dr. Day:

Specific gravity.....	0.8439 (35.9 Bé.)
Color.....	olive green
Begins to boil.....	80° C.
0° to 150° (0.7205 specific gravity), per cent.....	8.0
150° to 300° (0.7928 specific gravity), per cent.....	38.5
Residuum (0.9340 specific gravity), per cent.....	53.5
Paraffin, per cent.....	2.0
Asphaltum.....	none
Sulphur, per cent.....	0.2
Water.....	none

Analysis of oil from the Union Pacific well near Spring Valley, Uinta County, Wyoming. Analysis made by Wilbur C. Knight:

Specific gravity of crude.....	0.81 (44° Bé.)
Flash point.....	70° F.
Temperature given off—Degrees C.	Specific Gravity
77 to 130	0.7230 Oil at 15° C.
130 to 170	0.7540 Oil at 15° C.
170 to 200	0.7800 Oil at 15° C.
200 to 259	0.8040 Oil at 15° C.
259 to 292	0.8190 Oil at 15° C.
292 to 320	0.8340 Oil at 15° C.
320 to 350	0.8470 Oil at 18° C.
350 to 370	0.8580 Oil at 22° C.
370 to 380	0.8640 Oil at 22° C.
380 to 400	0.8880 Oil at 22° C.

The above analysis represents about the following composition:

Gasoline and lighter, per cent.....	20 to 30
Kerosene, per cent.....	30 to 40
Paraffin, per cent.....	10 to 20
Lubricating (residue).	

Gas in the Dakota Sand

A strong flow of very "dry" gas was found in the Dakota sand by a well drilled by the Canadian Geological Survey at Pelican Rapids on the Athabasca River in 1897–98. This flow has been continuous until the present time. The gas has a strong odor of sulphur, but no petroliferous smell whatever, nor has any condensation been noticed near the well head even in the coldest winters. It is evidently a very "dry" gas consisting principally of methane.

A considerable gas pool has been developed in the vicinity of Bow Island in southern Alberta; 17 or 18 wells have been drilled in the pool,

and are producing gas from the Dakota sand. Their total estimated capacity is something more than 75,000,000 cu. ft. per day. The gas has the following analysis:

	Per Cent.
CO ₂	0.00
CO.....	0.00
Oxygen.....	0.10
Heavy hydrocarbons.....	1.80
CH ₄ (methane).....	86.70
Hydrogen.....	5.40
Nitrogen.....	6.00

Recently a well drilled by the Edmonton Ad Club 5 miles north of Viking, Alberta, on the author's location, obtained a considerable flow of gas from what may be the Dakota sand. Although this gas is reported dry and odorless, the discharge from the well showed upon analysis a small petroleum content. The well was evidently drilled near the edge of the salt-water zone, as after blowing open for a week or 10 days, it began to make considerable brine. Analysis showed the gas to be high in methane, and of about 1,000 B.t.u. per cubic foot.

Gas and oil are produced from the Cloverly formation in the Big Horn Basin in Wyoming.

Salt Water in the Dakota Sand

No salt or fresh water has made its appearance in the Pelican government well as yet. However, as stated in a preceding paragraph, the Dakota sand may be saturated with salt water up the dip almost to the well where gas was struck at Viking, or approximately 100 ft. above sea level. The gas sand in this well may, however, be a sand body above the Dakota.

A well drilled at Brooks in southern Alberta found a strong flow of salt water in the Dakota at an elevation of 460 ft. below sea level. The Dakota sand in the Bow Island gas field is found at approximately 500 ft. above sea level.

Fresh Water in the Dakota Sand

This sand is an important artesian reservoir in South Dakota, eastern North Dakota, southern Manitoba, in the vicinity of the Black Hills uplift, in eastern Colorado, and in parts of Kansas and Nebraska. This fresh-water zone extends in Canada at least as far west as Moosejaw in Saskatchewan, and probably includes most of the central synclinal basin and the flanks of the North Saskatchewan syncline. The flanks of the Calgary Basin are probably saturated with salt water in the Dakota sand west of Bow Island and Viking. The reasons for this assumption

will be discussed later. Fresh water is associated with the oil in the Powder River pool and in the Moorcroft pool, on either side of the Sheridan Basin.

Gas is also found at places in South Dakota associated with fresh water in artesian wells.

Structure

The extensive faulting of the formations along the eastern edge of the Rockies is flanked by a great structural basin or trough, whose axis passes through Calgary (Fig. 1), at which point the Dakota would be found at approximately 2,300 ft. below sea level (5,500 ft. below the surface). East and north of this great structural basin occur three broad anticlinal folds:

a. The southernmost of these crosses the International Boundary in the vicinity of the Sweet Grass Hills in Montana (Sweet Grass anticline).

b. The central fold extends in a southwest direction from near Vegreville, crossing the Saskatchewan line near the 52d parallel (Battle River anticline).

c. The northeastern outcrop of the Cretaceous formations in Canada represents in a general way the axis of the northernmost of these three folds, and the one the attitude of whose sides shows the most gentle dips. This has been called by the author the Athabasca anticline.

It may be said that the prevailing dips encountered in the formations east of the axis of the Calgary Basin are remarkably gentle and uniform, and show practically no local irregularities or distortion due to violent secondary folding or torsional stresses. This has undoubtedly been an important factor (in connection with the "sheet" character of the Dakota sand) in the accumulation and redistribution of oil and gas in Alberta and western Canada generally. It will have to be taken into consideration in all prospecting for oil and gas in this region.

Southeast of these three great arches lies a broad, deep, structural basin (Moosejaw synclinorium), which occupies a large portion of North and South Dakota, northeastern Montana and southern Saskatchewan. The outcrop of the Dakota sand on the northeastern flank of this basin is nowhere more than 900 ft. above sea level. At the base of the Black Hills uplift it is about 1,000 ft. Its outcrop in the Black Hills is from 3,000 to 5,000 ft. On the flanks of the Big Horn Mountains its outcrop is from 4,000 to 6,000 ft., which is approximately true in the Lander district in Wyoming. It rises with the general uplift southward to more than 7,500 ft. above sea level at its outcrop in Colorado.

In a basin between any two of these uplifts the best chances for prospecting for oil in the Dakota would seem to be in the local domes and anticlines which may be found along the side of the basin where the Dakota outcrops at the higher elevation. This is believed to be the

case for the reason that there will be a constant movement of water from the upper side, issuing in springs along the lower side outcrop; while water entering from the upper side may do so locally, and, being under slight head, in seeking the line of least resistance to flow will avoid strong folds or domes. On the other hand, the strong hydraulic head existing along the side of the basin where the lower outcrops occur will tend to keep the sand saturated with encroaching water under pressure, which will usually issue at the low points of a vertically irregular or "scalloped" outcrop. However, the higher areas on such an irregular outcrop will be exposed to inward-seeping surface waters, which will tend to a movement of all water, and any possible oil which had risen to such points, to an exit at lower points on the outcrop where the hydraulic head from the opposite side of the basin is effective.

Thus, not only would "structure" be relatively less effective in retaining oil on the lower side of such a basin, but migrating oil carried with the water movement would be forced to the edge and lost by erosion and evaporation. On the higher side of the basin the tendency would be for oil to be carried down into the sand, and retained in irregularities in the structure.

The small quantities of oil found associated with fresh water along the lower side of such a basin will probably be of heavy gravity.

While the foregoing considerations apply to the Dakota sand, yet the best chances for developing production on the sides and edges of such a basin as that between the Black Hills and the Big Horn Mountains will probably be in overlying sands of a more lenticular shape, or at least having an outcrop on only one side of the basin. This is the case regarding the Shannon and Wall Creek sands in the Salt Creek pool.

Pressures

The rock pressure at the well located at Pelican on the Athabasca River is reported by H. L. Williams to be 225 lb. per square inch. As the difference in elevation between the top of the sand at this point and at its nearest outcrop at Boiler Rapids is very close to 500 ft., it will be seen that this corresponds very closely to that which might be expected on the assumption of Orton's hydrostatic theory. In this district the Dakota sheet sand under consideration offers almost ideal conditions for the application of this theory of underground pressures.

The outcrops of the Dakota along the Athabasca are also the nearest (and the lowest) to the well at Viking on the Battle River anticline, where what is supposed to be the Dakota was struck at approximately sea level. This gives a theoretical head of 1,060 ft., or an equivalent of 460 lb. pressure, which was the initial pressure encountered in this well.

When one comes to the Bow Island field, with initial pressures of

800 lb., the situation is complicated by the lack of knowledge as to seepage conditions in the Sweet Grass Hills, and also the density of the salt water in this district, which Orton apparently neglects. However, there is no important deviation from the pressures which might be expected in the Bow Island district on the assumption of a head of water from the Sweet Grass Hills. The hydraulic gradient would tend to dissipate a great part of this pressure (Darton: *Professional Paper No. 56, U. S. Geological Survey*) before reaching the Viking well, as outcrops of the Dakota are open on several sides of the Bow Island pool. This is based on the assumption of the gas-bearing formation at these two points being identical. The log of the lower part of the Viking well was badly kept, and the author has seen no results of the fossil evidence found. In case the gas-bearing sand is an upper lens, the above agreement still holds true, as in the flat muskeg country in the Athabasca Valley both formations outcrop on the same plane and at about the same elevation above sea level.

Geological History of the Dakota Sand

1. As has been shown, the Dakota sand was deposited over a wide area in shallow fresh water with prevailing strong currents. It was probably in part æolian.

2. There then followed the gradual sinking below sea level of the whole basin, but deepest in the central and western portion, as the Cretaceous sea covered the Great Plains area. Then followed the deposition of black marine shale deposits, with a few local sand bodies as brackish or fresh-water conditions might prevail locally for short periods. These deposits were very general over wide areas, as evidenced by the extent over which we find the Benton, Niobrara, and Claggett or Lower Dark shales, and the Clearwater shales of the Athabasca River.

3. As shown by King (*19th Annual Report U. S. Geological Survey*) and Roswell H. Johnson (*The Rôle and Fate of Connate Water in Oil and Gas Sands, Trans., li, p. 587, 1915*):

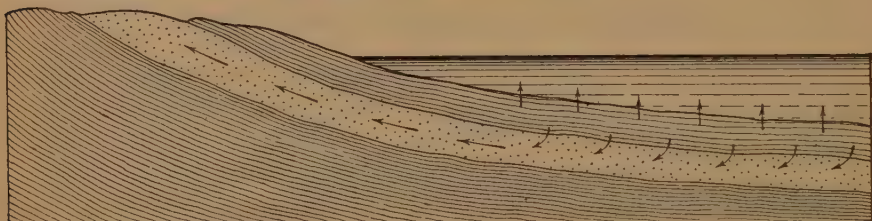
"As any particular sand body becomes weighted by a heavier and heavier overburden, the result of increasing deposition, a part of the large percentage of water in the uncompacted sand and mud is forced out. The water percentage of freshly settled mud is vastly higher than that of the resultant shales, or even of very porous sand. If beds of less compressible material meet or underlie the shales in question, there will frequently be a lateral motion along such beds to some point where upward movement will be resumed."

This is shown by King's diagram (Fig. 6).

On the assumption that most oils are formed in shales rich in organic matter in close proximity to sands which now contain these products, the source of oil (or gas) in the Dakota sand must have been the overlying

marine shales which are even yet so rich in carbonaceous matter. The unconformity upon which the Dakota was laid in the district in which it has been proved to be so rich in tar, that of the Athabasca region, presumably eliminates the lower formations in that district as a source of such hydrocarbons. The relative scarcity of organic life in a fresh-water sand such as the Dakota and the underlying fresh-water Comanchean (or Jurassic) makes doubtful in most districts the possibility of the formation of any great amount of hydrocarbons within the sand body itself, or within those beds immediately below it. It is more likely that oil found in such underlying sands had its origin in contiguous shale beds, or even possibly in shales overlying the Dakota itself, having migrated in some such manner as shown by King's experiments cited above.

In general, it is believed that no oil which has its origin in a lower formation will be found in a bed which has been deposited unconformably upon older and already consolidated rock deposits, unless after the later deposition some tectonic change takes place.



After King, 19th Annual Report, U. S. Geological Survey, part 2, p. 80

FIG. 6.—DIAGRAMMATIC SECTION SHOWING THE FLOW OF CONNATE WATER, OIL, AND GAS, DUE TO CONSOLIDATION OF SEDIMENTS.

4. As demonstrated by King, a portion of the water of deposition of shales overlying a coarse sand body is squeezed downward into the region of greater porosity, and passes laterally toward any outcrop where it may again have a chance to resume its upward movement. This movement of the connate water would therefore carry with it a considerable portion of any oil and gas which had been formed, segregation taking place selectively during its passage through regions of varying porosity.

During Cretaceous times the land surface to the northeast was being very gradually raised, offering an outcrop toward which movement of the liquid content of the Dakota took place. As pointed out by Roswell H. Johnson:

"Oil, gas, and water have extremely slight capacity for gravitational sorting while in a state of rest, but when moving the gravitational sorting is readily accomplished. As oil, gas, and water pass a body of larger pores, the gas, owing to its lack of capillary attraction, is retained in the larger pores." (*Rôle and Fate of Connate Water in Oil and Gas Sands, Trans.*, li, p. 587, 1915.)

Much oil was therefore probably carried with the water to the outcrop and from there lost by evaporation and erosion, leaving a relatively higher proportion of gas remaining in the sand along with the residual water. Where the escape and erosion were not so rapid as on the north-eastern front, oil would be forced by this movement out into the periphery of the basin and there retained, the deeper central portions of the Dakota sheet being occupied entirely by water (possibly salt) and some gas.

In regions of cross-bedding and lenticular deposition of the sand, of course such movement and gravitational sorting would be complicated by other barriers.

5. After such distribution of the fluid content of the shales and sands had been effected, in late Cretaceous time came the Black Hills uplift, the Big Horn uplift, and the upward tilting of the whole southwestern portion of the sedimentary blanket underlain by the Dakota sheet sand, in which already a considerable concentration of hydrocarbons had taken place. This tilting had an enormous effect on the hydraulic conditions in the Dakota sheet. The whole northeastern front had by this time been exposed by erosion, and fresh water had probably entered through seepage from this outcrop, already tending to force any residual oil body toward the lowest point of the "V" between the higher fresh-water sealed outcrop and the northern edge of the sand lens between the Athabasca and the Peace Rivers. Then, when the great southern and western uplift took place and a large proportion of the reservoir was raised above the general level, and folding formed the Calgary and Moosejaw synclines and their corresponding folds, this hydraulic head caused a great flood of water through the Dakota northward, probably carrying with it some of the remaining oil. The synclinal basins were already saturated, so to again establish equilibrium, the excess water was forced out by way of the outcrops to the north and east. By this scouring action in the sand, according to the principle of "selective" segregation, the dry gas clung to the more porous portions at the crest of the structural arches, the water spilling across the structural saddles or divides.

The remaining portion of the oil carried by the water was then either (a) forced to the outcrop and lost; or (b) forced into closely overlying sand lenses or crackled shale; or (c) forced out into the region where the Dakota is still under cover and of less uniform porosity—that is to say, along the boundary of this sand body, where more or less isolated lenses of sand offered catchment for such oil. Such lenses as these, however, probably already contained oil and water which had been left relatively undisturbed by such movements.

The remains of the action first mentioned are seen in the extent of the northern Athabasca tar sands, and the frequent fragmentary remains of the tar-saturated sand found with the glacial drift on the plains of

Alberta south of the outcrop of the Dakota. It may be mentioned that the lightest oil found in this tar-sand district has a specific gravity of 1.02, as shown in a previous paragraph.

The probabilities of the second action do not increase the promise of finding large oil pools or reservoirs in such shales or minor splits of the Dakota overlying the main sand body, but rather will be indicated by paraffin shales or by thick, viscous water-washed tar occurring near the top of the sand, such as that found in the hole drilled at Pelican Rapids, in the upper part of the gas-bearing Dakota.

The possibilities of the third action mentioned above offer the chance of developing important oil pools in the Dakota sand in certain areas whose geological limits are already approximately known, lying west and south of the Athabasca River. In the southern part of the Dakota area, in the United States, in parts of Montana, Wyoming, Nebraska, and Colorado, where the area underlain by this sand is much disturbed and folded by mountain forming, there are of course many unprospected areas for oil and gas in the Dakota. These represent pools of oil and gas cut off by varying porosities, by intrusions and accompanying folding, from reaching the outcrops of the sand, as in the Big Horn Basin, and the Salt Creek and Powder River pools.

In this southwestern part of the Dakota Basin the conditions of deposition were more irregular, and there were probably a number of islands in this part of the Cretaceous sea. These conditions, combined with later folding, had a tendency to limit the movement of fluids in this sand to local areas and lenses, which have already resulted in the development of good oil and gas pools. The relative time of folding is a factor which is often neglected in considering the oil and gas possibilities of a formation in certain districts.

With reference to the Moose Mountain district in Alberta, and the series of sharp folds south and west of Calgary, it will be noticed that there is a considerable thickening of the Dakota series along the mountains, to approximately 1,000 ft., composed principally of quartzose sandstone and shale beds, with the basal conglomerate bed from 30 to 60 ft. in thickness.

As the great Cretaceous mediterranean sea deposited layer after layer of marine ooze and mud as an overburden to the porous Dakota sand, this former shore line of the Dakota Basin constituted a very open outcrop for the upward movement of circulating fluids continually being forced into this bed from overlying compacting shales (Fig. 6). Moving laterally along the sand, this water suddenly entered the thickened section at the upward-bent shore line. Such conditions probably established a sharp pressure gradient within a short distance, which may account for the deposition by ascending waters of the siliceous cement which caused the quartzitic condition of the Dakota along this western outcrop.

While these conditions may have sealed an outcrop locally, yet in general the thickness of the sand along this belt presumably offered too open a condition to permit of any considerable accumulation of oil back of a generally sealed edge. If this hypothesis is correct, the chances of finding oil in commercial quantities in the Dakota in that region are greatly lessened.

Summary

1. In general, on account of the very uniform character and thickness of the Dakota sand over wide areas, it is not believed to be a prospect horizon of the first class for the development of oil production, except in certain limited districts.

2. In Canada it is believed that the most favorable areas for the development of oil production in the Dakota sand will be found in the region underlain by this sand where it begins to play out and become discontinuous and lenticular in its nature. This is the case in the oil pools of Wyoming producing from the Dakota sand, and points to certain parts of Montana as worthy of prospecting.

3. While not a prospect of the first order, the testing of the Battle River anticline for oil in the Dakota is distinctly worth while. This anticline is, for the foregoing reasons, more favorable for gas in this sand; and oil should be looked for, if at all, in the more laterally restricted sands occurring in the Benton or Niobrara, or higher.

4. The chances for the development of oil production in the formations lying below the Dakota in western Canada, are not of the first class from a commercial standpoint.

5. Where structural and other conditions are favorable for the concentration of oil, the relatively more lenticular sand bodies above the Dakota, particularly in the Benton and Niobrara, are more promising for the development of production than is the Dakota sand itself; except in such regions as mentioned in the preceding paragraphs.

*Logs of Alberta and Montana Wells**Pelican Oil & Gas Co.'s Well No. 1, Pelican Rapids, Northern Alberta.*

(Dec. 3, 1912. Well Head 1,300 ft. Alt. approximately)

Formation	Top	Bottom	Formation	Top	Bottom
Blue and yellow shale...	1	66	Dark gray shale, sandy..	767	843½
White and gray shale			Sandy shale.....	843½	872
(water).....	66	82	Sandy shale.....	872	882
Blue shale.....	82	200	Coarse rock mixed with		
Blue and brown shale..	200	235	heavy oil.....	882	887
Brown shale.....	235	285	Shale and sand.....	887	898
Gray-brown shale.....	285	331	Hard rock.....	898	903
Sand rock (hard).....	331	352	Lime carrying oil.....	903	997
Shale.....	352	365	Limestone.....	997	1,051
Sand rock.....	365	425	Hard flinty shell (strong		
Shale.....	425	507	flow gas under shell)...	1,051	1,053½
Brown shell (hard).....	507	509½	Limestone.....	1,053½	1,158
Gray shale.....	509½	538	Hard lime shell.....	1,158	1,159
Shell.....	538	540	Limestone.....	1,159	1,192
Sandstone.....	540	546	Hard shell (gypsum)....	1,192	1,197
Shale.....	546	575	Blue shale and gypsum..	1,197	1,293
Hard shell.....	575	581	Hard lime shell.....	1,293	1,296
Gray shale, streaks			Lime rock.....	1,296	1,538
sandstone.....	581	644	Lime, shale, and lime		
Strong flow gas.....	625	rock.....	1,538	1,560
Gray shale (gas).....	644	653	Gray shale and lime (gas)	1,560	1,700
Gray shale (soft and			Limestone.....	1,700	1,784
cement-like).....	653	666	Hard shell.....	1,784	1,790
Sand rock.....	666	671	Limerock (shale streaks)	1,790	1,875
Gray shale.....	671	688	Hard shell.....	1,875	1,879
Hard brown shell.....	688	689	Layers of limestone and		
Dark gray shale.....	689	740	shale.....	1,879	2,040
Hard shell.....	740	741	Strong flow of gas.....	2,040
Dark gray shale.....	741	766	Limestone and shale, in-		
Hard shell.....	766	767	terstratified.....	2,040	2,069

Morinville Well No. 1, drilled by H. L. Williams, near Egg Lake, Alberta.

Formation	Top	Bottom	Formation	Top	Bottom
Surface drift with clay boulders.....	0	250	Gravel (salt water)....	2,900	2,902
Sand rock.....	250	260	Greenish shale, like "dobe" shale.....	2,902	2,940
Blue and brown shale, thin layers of sandstone.....	260	440	Sandrock (oil), flow gas underneath.....	2,940	3,052
Sand rock with gas....	440	465	Hard shell with iron....	3,052	3,064
Blue and brown shale with one or two thin layers of sandstone....	465	1,410	Greenish shale, like "dobe" shale.....	3,064	3,100
Dark blue shale with heavy oil seep.....	1,410	1,415	Blue shale with thin layers of sandstone....	3,100	3,200
Blue shale, dark blue to light blue and green....	1,415	2,450	Greenish shale, very sticky (dobe).....	3,200	3,260
Hard ironstone shell....	2,450	2,456	Hard ironstone shell....	3,260	3,262
Blue and gray shale with gas.....	2,456	2,900	Blue sandy shale with oil seepages.....	3,262	3,310
			Blue shale, interstratified with hard lime shells...	3,310	3,340

From 3,040 to present depth, more or less oil, increasing. Shale from 3,040 very hard, with pyrites, but slacked on exposure.

Fort McMurray—Great Northern Exploration Co. Well No. 2.

(Finished August, 1912.)

Formation	Top	Bottom	Formation	Top	Bottom
Surface (soil).....	0	17	Lime.....	502	562
Limestone.....	17	77	Shale.....	562	592
Shale.....	77	92	Lime.....	592	604
Lime.....	92	152	Salt (salt-water).....	604	704
Shale.....	152	192	Limestone.....	704	779
Soft shale.....	192	197	Salt.....	779	869
Lime.....	197	237	Lime.....	869	999
Shale.....	237	242	Shale.....	999	1,059
Lime.....	242	362	Brown sandstone.....	1,059	1,119
Shale.....	362	382	Brown "Medina" rock	1,119	1,139
Lime.....	382	462	Red rock, streaked, hard	1,139	1,405
Shale.....	462	502			

Viking Gas Well, 5 miles due north of Viking, Alberta.

Formation	Top	Bottom	Formation	Top	Bottom
Yellow clay.....	0	30	Sandy white shale.....	465	485
Blue clay.....	30	40	Brown shale.....	485	515
Gray sandy clay (little water).....	40	50	Grey shale (gas est. 50,000 ft.).....	515	530
Blue clay.....	50	75	Brown shale (gravelly and caving, gas).....	530	535
Shale.....	75	85	Brown sand (gas).....	535	585
Shale.....	85	100	Brown shale.....	585	655
Sandy shale.....	100	120	Gray sandy shale (water)	655	680
Loose sandy shale (water)	120	140	Brown shale.....	680	690
Blue slate.....	140	165	Water sand (slightly salt)	690	700
Light brown shale.....	165	170	Gray sand (salt water).	700	740
Gray shale.....	170	190	Blue shale.....	740	890
Gray sandy shale.....	190	210	Shell.....	890	895
Gray shale.....	210	230	Brown shale.....	895	1,210
Light brown shale.....	230	243	Hard lime shell.....	1,210	1,215
Coal.....	243	245	Blue shale.....	1,215	1,400
Dark brown shale.....	245	252	Brown shale.....	1,400	1,605
Gray sandstone.....	252	272	Sandy gray shale.....	1,605	1,610
Gray shale.....	272	280	Sandy shell.....	1,610	1,615
Brown shale.....	280	340	Brown shale.....	1,615	1,700
Sandy shale shell (water)	340	345	Bastard sand and shale	1,720	1,770
Brown shale.....	345	360	Brown shale, (gas in quantity).....	1,770	2,180
Gray shale.....	360	370	Blue shale.....	2,180	2,335
Brown shale.....	370	390	Hard capping, boulder formation (gas in quantity).....	2,335	2,340
Hard shell.....	390	395			
Brown shale.....	395	403			
Sandy shell (gas and water).....	403	405			
Gray shale.....	405	465			

Montana Oil & Development Co. Well, near Shelby, Mont.

(Drilling started July 16, 1912)

Formation	Top	Bottom	Formation	Top	Bottom
Black shale.....	40	160	Light shale.....	1,150	1,195
Lime shell.....	160	162	Hard shell.....	1,195	1,200
Black shale.....	162	315	Hard sand.....	1,200	1,223
Water and gas sand.....	315	320	Shell.....	1,223	1,230
Hard shell.....	320	320	Sand.....	1,230	1,300
Shale.....	320	420	Light shale.....	1,300	1,360
Gas sand.....	420	430	Sand.....	1,360	1,390
Sandy shale.....	430	460	Hard shell.....	1,390	1,400
Hard shale.....	460	470	Light shale.....	1,400	1,460
Black shale.....	470	650	Hard sand.....	1,460	1,500
Gray sandy shale.....	650	720	Hard shell.....	1,500	1,510
Black sand.....	720	730	Hard sand.....	1,510	1,550
Sandy shale.....	730	770	Yellow shale.....	1,550	1,600
Light shale.....	770	850	Gritty sand.....	1,600	1,650
Sandy shale.....	850	950	Hard shell.....	1,650	1,655
Black shale.....	950	1,045	Black shale.....	1,655	1,675
Gray sand.....	1,045	1,065	Hard shell.....	1,675	1,680
Gas sand (some gas)....	1,065	1,070	Lime rock.....	1,680	1,730
Black shale.....	1,070	1,100	Black shale.....	1,730	1,755
Light shale.....	1,100	1,115	Bottom of hole—		
Red rock.....	1,115	1,150	abandoned.		

Vegreville Well No. 1, Vegreville, Alberta

Formation	Top	Bottom	Formation	Top	Bottom
Surface.....	0	15	Bottom sand.....	1,362
Blue mud.....	15	272	Brown shale.....	1,362	1,440
Brown shale.....	272	325	Blue mud.....	1,440	1,558
Gas.....	328	Little gas.....	1,563
Shell 2 ft. thick.....	330	386	Brown shale.....	1,563	1,740
Water.....	386	Blue mud.....	1,740	1,868
Sand.....	386	410	Sand (gas 1,870).....	1,868	1,872
Blue mud.....	410	515	Blue mud.....	1,872	1,880
Gas.....	520	Brown sand.....	1,890	1,920
Blue mud.....	520	1,000.3	Blue mud.....	1,920	2,000
Sand.....	1,000.3	1,356	Bottom of hole.		
Gas.....	1,360			

DISCUSSION

E. W. SHAW, Washington, D. C. (communication to the Secretary*).
—The comparatively great extent and uniform character of the Dakota sandstone make it of great interest to the student of rock pores and the movements of their fluid contents. However, compared with other sands in areas of carbonaceous rocks facts concerning it do not seem to be of such great direct importance to the producer for it has thus far yielded few important pools of oil or gas. Information concerning the sandstone is extensive but scattered, hence a general treatise regarding it—particularly its pores, the fluids they contain, and the movements of the fluids—has been much needed. I therefore feel that the chief thing to be desired concerning Mr. Huntley's paper is that it were more extended, particularly as to the discussion of the data presented. The general structure map of the Dakota is of especial interest.

Slight changes in wording seem to be desirable here and there in the paper. For example, at the beginning of the third paragraph one wonders if flora instead of fauna were not intended. In Figs. 2, 3, 4, and 5 the vertical scale seems incorrect. Concerning the analyses of oil quoted it is noted that they "are given, as indicating what may be expected in different localities." This seems to need modification for the average analysis of oil from Powder River, which indicates for example that this oil begins to boil at 213° C. The fact is that the samples to which this figure refers were taken from seeps long exposed to the air and hence the underground Dakota oil of this district no doubt differs profoundly from that which the analysis indicates. The product of a well here would probably differ more in boiling point and other respects from the samples analyzed than it would from almost any other petroleum in the world. The use of the word synclinorium does not conform to the original definition, but this may be pardonable because misuse of the term, though a different misuse, has become general. Synclinorium is an orographic term and cannot be correctly applied outside of mountain regions, for it is derived in part from the Greek word for mountain. It has, however, been widely used in a purely structural sense for compound syncline. The broad shallow syncline of the Dakota sandstone falls in neither category. Objection might also be made to the use of "isobaths" (lines through points of equal depth) to show altitudes. The argument in the upper half of p. 340 does not seem clear and any possible interpretation of the words seems to involve fallacious ideas.

One of the first points made by Mr. Huntley is that the Dakota rests unconformably on various underlying formations, and the signifi-

* Received Oct. 28, 1915.

† Published by permission of the Director, U. S. Geological Survey.

cance of this point is apparently brought out at the top of p. 342 where it is stated that the "unconformity presumably eliminates the lower formations in that district as a source of such hydrocarbons." This point is of great interest because some investigators believe that unconformities play an important part in oil and gas accumulations by checking the rise of such fluids from below, though the fluids may be found both above and below the unconformity. To me it has seemed probable that oil and gas are much more frequently the product of downward concentration than is generally supposed and with regard to the Dakota in particular I am inclined to believe that Mr. Huntley is correct in his inference that oil has been carried downward by circulating water and has accumulated against the unconformity, the water passing on and the oil remaining behind in places where the pores or other openings in the rocks are large and the structure favorable. If this idea is valid, then in a sand of comparatively uniform porosity, which outcrops all about its border but is considerably higher along one side than the other, water should flow freely from the higher side of the border to the lower and might be expected to wash out most of the oil, gas, and salt water, so that the sand would contain and yield large amounts of fresh water compared with other sands. These inferences seem to be applicable to the Dakota sand and I have thought of it as one which might have been very productive of oil and gas had not circulation been so free and uniform throughout its extent, the reason for the free circulation being its uniform porosity, extensive outcrop, and the fact that one edge is considerably lower than the other.

The fact that the sand as a whole has a basin form does not seem to affect the results materially, because the basin is not so deep but that the hydraulic pressure along its higher or western edge is effective in moving the fluids throughout practically its entire extent. If the basin were deeper, or the pores on the whole smaller, the effect of the greater pressure along the western edge might be to cause most of the flow to take place about the north and south ends of the basin and if this had occurred the salt water, oil, and gas might not have been so completely washed out of the central parts of the basin. Again, if the porosity of the rock had been more variable it would seem that many more pools of oil and gas might have been retained in parts of the sand where the pores were larger, particularly where sand with such large pores occurred on anticlines.

If the source of the oil and gas had been below the Dakota sand the results due to porosity, general eastward tilt, and extent of outcrop, might not have been very different, but under the hypothesis that oil and gas pools accumulate on anticlines from surrounding "gathering ground" without the aid of circulation, the results would have been of very different sort. The fact that far more carbonaceous material occurs above

the sand than below, and that probably still more has been removed by erosion, and the fact that apparently no unreasonable assumptions are involved in the hypothesis that oil and gas have been carried downward into the sand, have inclined me to believe that such a process has been operative, and that had it not been for the free and extensive circulation of water through the Dakota sandstone, the rock might have contained many important pools of oil and gas. The fact is, however, as Mr. Huntley points out, that the sandstone is not altogether uniform in porosity but only comparatively so and that it has in places comparatively steep dips—spots which water could more easily flow around than across—hence there are places in the formation where oil and gas have accumulated, and may now be recovered.

At the bottom of p. 331 the fresh water of the Dakota throughout large areas is attributed to lakes and musky swamps on a part of its western outcrop, "drainage conditions having kept its exposure more or less saturated with entering surface water since Cretaceous time." I think the Dakota would have contained fresh water if there had been no lakes or swamps on its outcrop and I know of no reason for supposing that the lakes and swamps referred to have existed for more than a very small part of the time that has elapsed since the Cretaceous period. The fresh water seems more reasonably attributable to the general permeability of the sand and its general dip eastward to an extended outcrop.

Although I am unable to follow and agree with all of the arguments in Mr. Huntley's paper, the conclusions stated in the summary, partly for reasons not set forth in the paper, seem in all probability correct. The fourth conclusion in particular is worthy of emphasis and elaboration. If, as I have long been inclined to believe, the source of most, if not all, of the oil of the region is above the Dakota, the oil once within the sand would tend to rise as high as possible in it and no matter what lateral movements took place later, there would be little chance of migration into lower sands where their beveled edges touch the Dakota. Another condition favoring the same result is the finer-grained character of the lower sands. On the other hand, if the prevalent view that the source of the oil is below the sand is correct, then pools should have been accumulated along the route of the upward journey of the oil in crevices and bodies of open sand surrounded with material having smaller pores. Once the oil is in such a sand as the Dakota, it would seem to matter little whether its tendency is to move up or down the dip; little or none should leave the sand for a lower stratum unless a considerable body of sand became completely filled with oil.

Correlation and Geological Structure of the Alberta Oil Fields*

BY D. B. DOWLING, † B. AP. SC., F.R.S.C., OTTAWA, ONT., CANADA

(San Francisco Meeting, September, 1915)

THE interest which has been aroused in prospecting for oil in the foothills of southern Alberta, and in the oil possibilities of the known gas fields situated in the less-disturbed areas, called for a much closer examination of the structure, thickness, and composition of the underlying rocks of the region than had hitherto been made. The areal geology of the larger part of the great plains was outlined by Dawson, McConnell, and Tyrrell, between 1881 and 1885. The foothill area was not critically examined at that time, owing to the time which would have been required for its proper study and the difficulty that was found in recognizing in the foothills the divisions which had been adopted in the mapping of the formations of the plains. This was due in great measure to the paucity of exposures in continuous sections of the lower divisions of the Upper Cretaceous. Since the pioneer work on the plains was published, the beds which form continuations south into Montana have been critically examined. The Canadian beds arch over the end of a flat anticline which rises to the south; and two sections in Montana, one on each side of the anticline, help to explain some of the difficulties previously encountered in the mapping of the measures found in the foothills. The index map, Fig. 1, shows the area studied, and its relation to southeastern Alberta, and to Montana.

Problems of Correlation

The principal difficulty experienced in correlation was in connection with the measures known as the Montana group. The beds of this group did not seem to agree in the two Montana sections. They showed the same differences to some extent as are encountered in comparing the foothill measures with those of the plains.

The Montana group in Dakota and Manitoba consists of a series of shales of marine origin. In the eastern Montana section two sandstone members appear, leaving a shale member at the top and another

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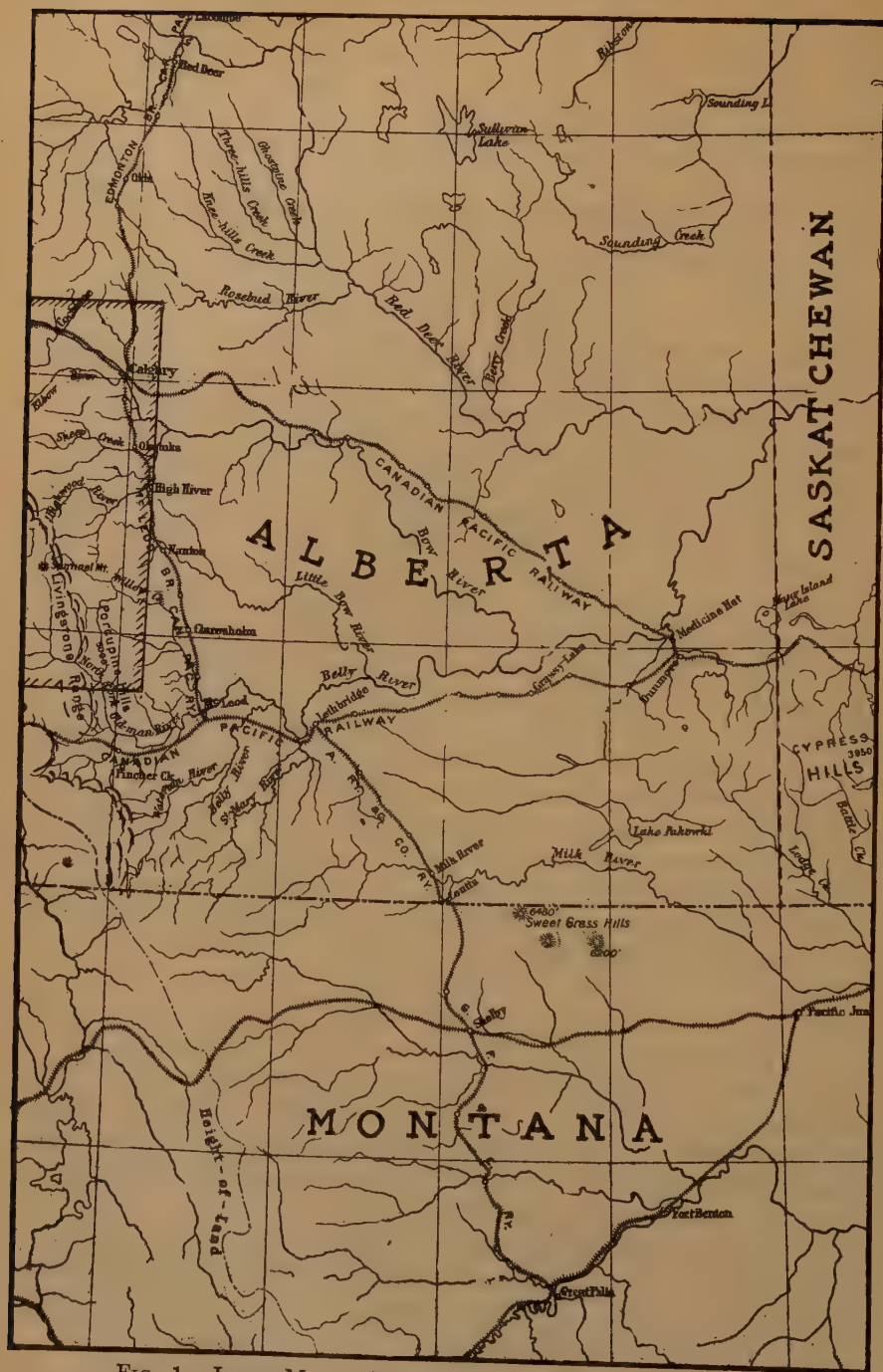


FIG. 1.—INDEX MAP OF PORTION OF MONTANA AND ALBERTA.

near the bottom. In the section in western Montana the lower shale is not found and the upper shale is reduced in thickness (Fig. 2).

The eastern and western sections have been correlated by tracing them to the International Boundary and making a connection by means of the formations on the Canadian side.



FIG. 2.—IDEALIZED WEST-EAST SECTION OF CRETACEOUS SEDIMENTS NEAR THE CLOSE OF THE MARINE INVASION.

The nomenclature adopted for the divisions of the Upper Cretaceous in various parts of western Canada and in the United States is given in the accompanying table.

These formations are found in the foothills, and it is evident that in any section that will be examined in the Montana foothills west of Cutbank we may expect the probable absence of the upper shale member and the appearance of additional sandstone members in the shales of the Benton beneath. Sections to the east will probably show a gradual diminution of the sandstone members. As the area under consideration lies to the north similar variations in the deposits no doubt occur and a few notes on the sections there observed are here introduced.

Dakota Group

The Dakota sands, which contain gas at various points, heavy asphalt oils on the Athabaska and some light paraffin oils in the vicinity of Calgary, are probably represented in the Sweet Grass Hills by one of several sandstone beds which apparently lie at the base of the Benton. To the west these sandstone beds thicken as the mountains are approached; and in the Blairmore section on Crowsnest River the basal beds lying above the coal measures of the Kootenay are upward of 2,000 ft. in thickness. To the north they are less than 1,000 ft. thick in the foothills west of Calgary, and decrease to 200 ft. on the Athabaska at some distance from the mountains.

Colorado Group

The Colorado group, mainly shales with an occasional sandy member, underlies probably the whole of the plains and is found inside the mountains. Its thickness in the Sweet Grass Hills is estimated to be 850 ft., and in the vicinity of the mountains at Blairmore 2,700 ft. In the foothills it may well be more than 2,000 ft.

	Western Montana	Southern Alberta			Central Montana	Manitoba	South Dakota
		West Forks of Milk River	East near Pakowki Coulee				
Montana Group	Bearpaw shales (<i>Marine</i>)	Foxhills Pierre	Bearpaw shales (<i>Marine</i>)	Bearpaw shales (<i>Marine</i>)	Bearpaw shales (<i>Marine</i>)	Odanah shales (<i>Marine</i>)	Foxhills
	Two Medicine formation (<i>Fresh and brackish</i>)	Belly River formation	"Pale" and "Yellow" beds Sandstones and clays (<i>Fresh and brackish</i>)	"Pale" and "Yellow" beds Sandstones and clays (<i>Fresh and brackish</i>)	Judith River formation (<i>Mainly fresh water</i>)		Pierre
			Shale at Forks	Shale in coulee (<i>Marine</i>)	Claggett shales (<i>Marine</i>)	Millwood shales (<i>Marine</i>)	
	Virgelle Sandstone		"Castellated" rocks	"Castellated" rocks of Milk River	Eagle sandstone		
Colorado Group	Benton shales (<i>Marine</i>)	Lower Dark shales (<i>Marine</i>)			Benton shales (<i>Marine</i>)	Niobrara shales (<i>Marine</i>) Benton shales (<i>Marine</i>)	Niobrara shales (<i>Marine</i>) Benton shales (<i>Marine</i>)
Dakota Group		Exposures of sandstones in the Sweet Grass Hills, and also reached by drilling in Alberta.			Exposures in Sweet Grass Hills		

Montana Group

The character of the deposits of the Montana group, as shown in the table above, indicates that the sea margin changed several times but remained during this period within the area now occupied by the foothills and the plains, and that in tracing these beds eastward a diminution in the thickness of the sandstones and an increase in the shales may be looked for. Northward the problem of the variations in the deposits is being studied, and a few points that have been determined may be mentioned here.

It is fairly well established in the section between Medicine Hat and the mountains that the sandy beds at the base are very thin at Medicine Hat and in the foothills resemble the "castellated rocks" of the southern section. Northward they evidently become thin again, as they do not appear in the Athabaska section and may be represented in the mountains by the Brazeau formation or the Dunvegan sandstones of Peace River. The marine shales, called "Claggett" in the eastern Montana section, lying between the sandstone members, are probably represented at Medicine Hat by a much thicker shale series. In the foothills near the Crowsnest River this shale series has thinned out and cannot be recognized. It is probably present in the sections north of the Highwood River, increasing in thickness to the northeast, while the sandy beds of the underlying lower part of the Belly River series decrease materially and ultimately thin out. This shale would then, in the absence of lower sandstones, represent the series overlying the Colorado shales of the Athabaska River and of the northern foothills. The upper sandstone and clay member of the Belly River formation is thicker in the west than in the east and the section west of the Porcupine Hills resembles that in western Montana in the inclusion of all the Belly River rocks in one thick sandstone member.

The Bearpaw shale, the upper part of the Pierre, is a marine deposit that overlies the Belly River beds. It is exposed over large areas in southern Alberta and in a somewhat diminished form is found in the foothills to the south. Its thickness is apparently maintained westward; but the marine portion is confined to the central part of the section. The beds at the base, in the vicinity of Lethbridge and even farther east, contain coal-bearing zones, which in places have valuable coal deposits. In the foothills, coal seams are found both at the top and the bottom of these shales, and it is suspected that north of Highwood River the marine deposits of the formation are very thin if they are present at all; so that the division between the Edmonton sandstones and the "pale beds" of the Belly River formation may be marked farther north by merely a coal-bearing zone or by transition beds, instead of marine shales. Some indication of this is given in the character of the rocks in

the section on the Saskatchewan River east of Edmonton. The sandstone and clays of the Edmonton formation do not end abruptly, but merge by degrees downward into a shale series. The shales overlie sandstones which may possibly represent the top of the Belly River series, there composed of a thin sandstone member. The shales beneath are no doubt directly continuous with the Pierre of the east.

Foothill Sections

The geologist working in the foothills of central Alberta must therefore be prepared to find, not the well-marked formations so far discussed as found in other places, but, for the higher beds, a great series of sandstones, ranging in age from the earlier deposits of the Montana group, up to Tertiary deposits. There may be but traces of the marine beds which are found in the sections on the plains to the east. It may be noted also that the Benton formation is well developed, and forms the great shale series of the foothills, but that north of a line drawn northwest from the forks of the Highwood River, shales belonging to the lower part of the Montana group cap the Benton.

Structure

The structure of the outer portion of the foothills has been partly mapped and a comprehensive view of its general character may be gained from the accompanying sketch and sections, Figs. 3 and 4. It will be seen from them that there was an uplift of the lower measures toward the mountains, accompanied by profound fracturing throughout the disturbed zone. Since the lines of fracture penetrate below the beds containing the possible oil supply the fault blocks have necessarily a limited oil-drainage area and do not afford promising ground for wells. Our field of study has been limited, therefore, to the eastern edge of the broken country in the hope of finding anticlines in close connection with the less-disturbed beds of the Alberta syncline which lies to the east. One of these on Sheep Creek is being thoroughly prospected and one well is now producing a small quantity of very light oil. Another in Township 23 west of Elbow River has yielded heavier oil.

In the country to the east of this broken area and the syncline indicated in Figs. 3 and 4, the beds are so slightly flexed that they seem at any one point to be almost horizontal. They are as a rule less consolidated than the beds near the mountains, and the rivers are deeply trenched. This river erosion is accompanied in nearly every case by a series of land slips, extending back for some distance from the banks. This already has been interpreted as faulting by several "experts" and an intricate structure showing anticlines and faults has been pictured, providing many "oil companies" with attractive prospectuses. There

is a wide anticline, however, in southern Alberta, between the outer foothills and the Cypress Hills at the eastern boundary of the province, which extends from northern Montana well into Alberta. This had already been the subject of investigation for a possible natural-gas reservoir; and the wells at Bow Island which supply Lethbridge and Calgary are located on it. Attention has again been called to it by the discovery of slight signs of oil in springs on the slopes of the Sweet Grass Hills in Montana, and several drilling rigs have been placed in the valley of Milk River and even at the Boundary line, on the flanks of the above-named hills. The borings in this vicinity will probably penetrate the sandstones of the lower part of the Belly River series, and also the Benton, before reaching the Dakota, from which there seems to be some chance that gas at least will be obtained. The thickness of the Cretaceous measures is here smaller than in the foothills, and very deep wells will not be necessary to test the ground.

A flat anticlinal structure is also indicated by the outcrop of the Belly River rocks in the eastern part of Alberta. This anticline runs in a northwest direction and is crossed by several stream valleys, notably that of the Battle River. The Grand Trunk Pacific Railway crosses the Battle River near the axis of the anticline. A well, sunk for gas near the railway but to the west of the center of the anticline, struck a small gas reservoir at a depth of 2,340 ft.

Development

Oil seepages have been known for many years in the mountains along the International Boundary east of the Flathead Valley. Several companies bored wells at the outer edge of the mountains, and about six years ago there was some excitement over the discovery of oil in a well near the Waterton lakes. The difficulty of getting machinery to this region and the probability of the area being limited prevented extensive prospecting. The finding of oil last year in an easily accessible area of less broken country at the outer edge of the foothills at once attracted the attention of the speculative element of the population; and many companies were formed and oil leases applied for. The discovery well is situated on an anticline of Benton shales, flanked on both sides by sandstone ridges cut through by the valleys of three streams. Since the sandstone at the crown of the anticline has been removed by denudation, the direction of the anticlinal axis is marked by a series of transverse valleys eroded in the shales. These depressions afford favorable locations for derricks; and 11 wells are now being bored. In the country to the west of this anticline many other drillings are being made, so that in the portion of Alberta shown in Figs. 1 and 3 there were during 1914 about 36 separate points of attack, mainly in the foothill belt. Two wells

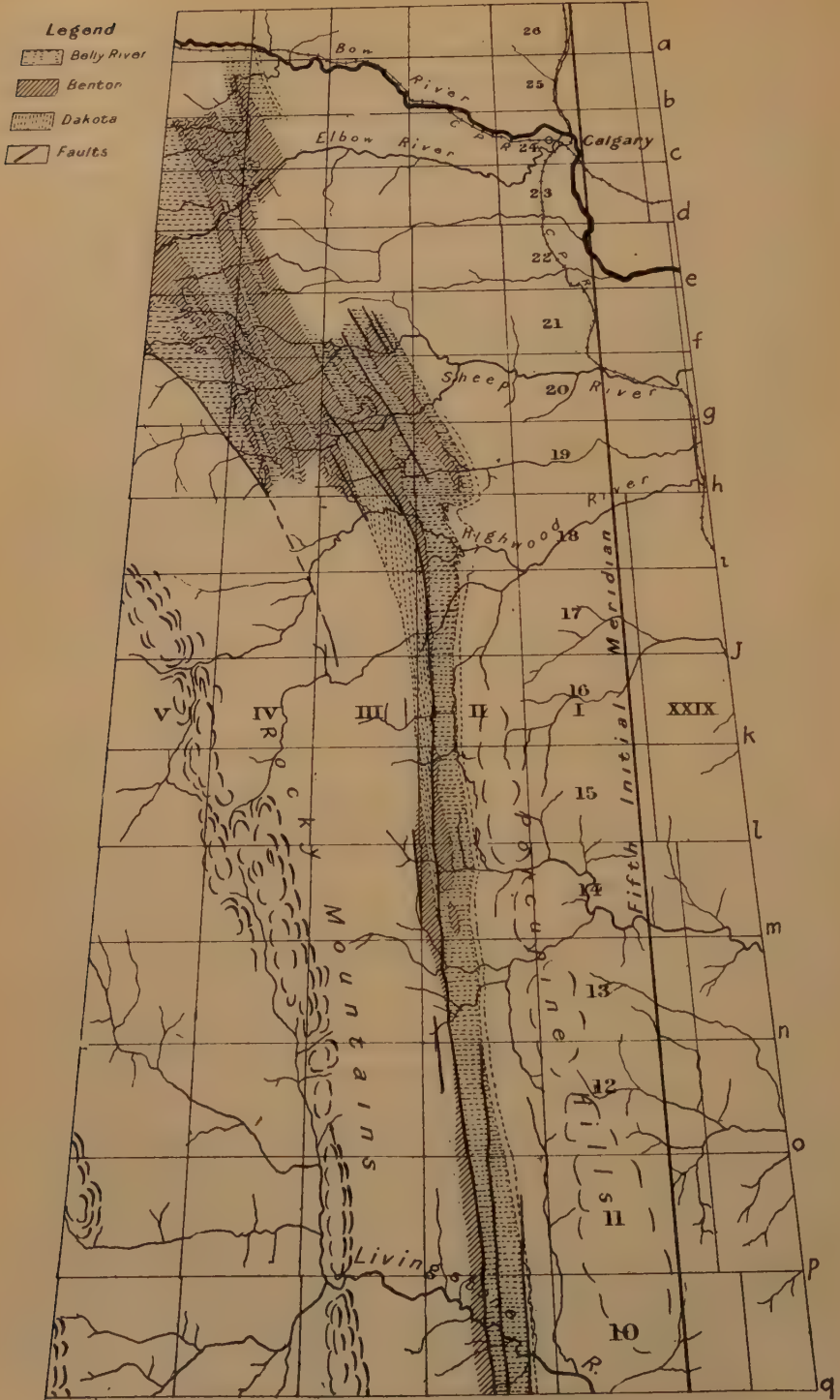


FIG. 3.—PERSPECTIVE DIAGRAM, FOOTHILLS OF SOUTHERN ALBERTA.

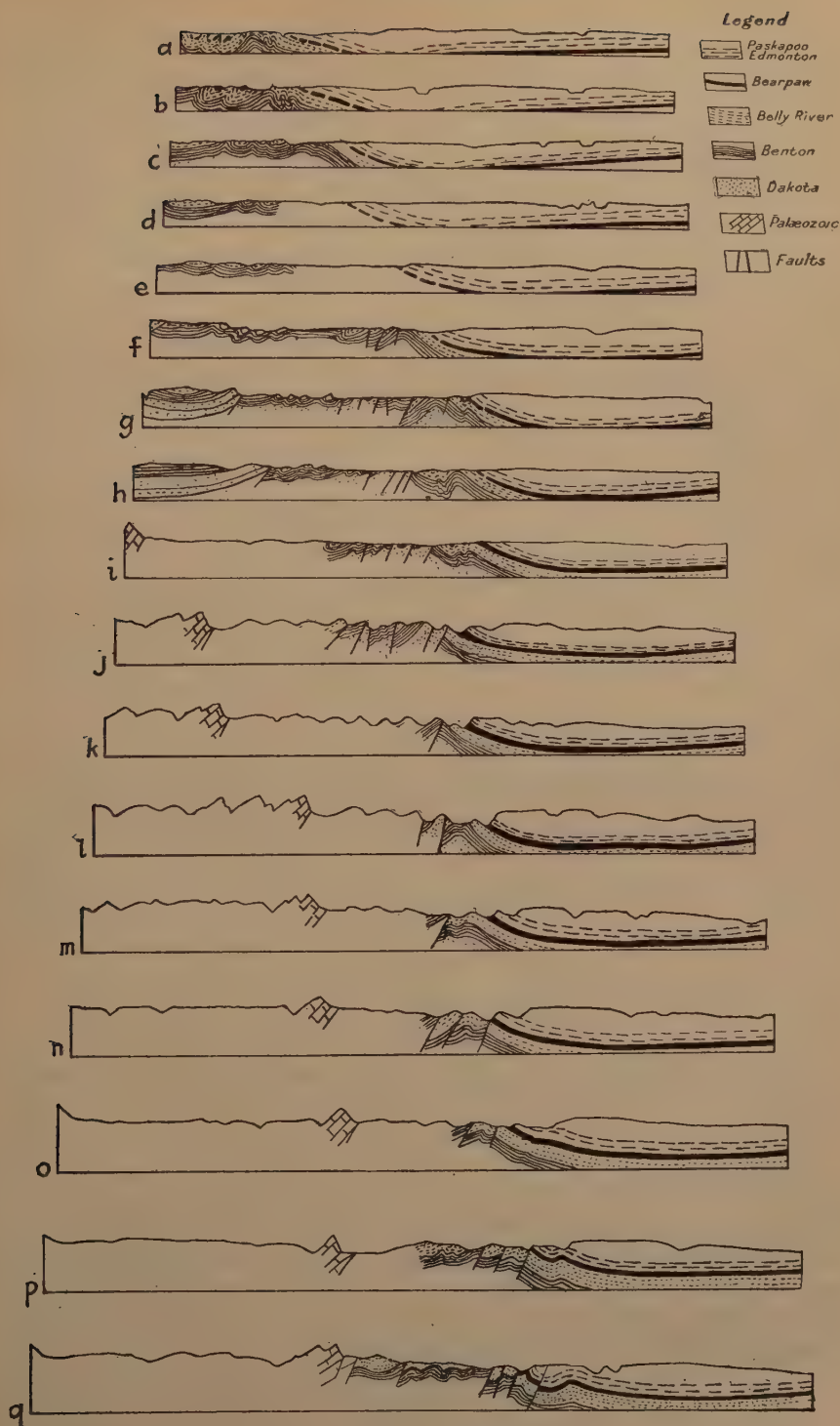


FIG. 4.—STRUCTURE SECTIONS.

have reached depths of more than 3,000 ft. without success. Seven, including the discovery well, are over 2,000 ft. deep. Fourteen are over 1,000 ft. deep and thirteen others have reached smaller depths.

Three companies are boring in the Milk River Valley and four in the foothills north of Bow River. In a few cases it may be considered that the ground has been found to be barren of productive reservoirs; but in the majority of cases the mechanical difficulties have been so great, owing to the depth to the prospective oil sands, that no positive result has been reached. In some cases the wells have been badly located from the viewpoint of structure.

In the discovery well, light gasoline oil and a heavy gas flow was found at 1,550 ft., in sandy beds in the lower Benton. At 2,700 ft. another flow of gas and oil was found in the Dakota or in sands of about that horizon. This oil was also light in specific gravity (about 55° Bé) and was accompanied by a heavy flow of gas which has been shown by experiment to produce a light gasoline on condensation.

It is claimed that showings of oil have been got in several wells in the vicinity.

A discovery of oil 40° Bé in the well of the Moose Mountain Oil Co. was announced on Nov. 24, 1914. The well has since been shot and a yield of 25 bbl. per day is claimed. The oil is dark brown and shows a greenish color by reflected light. It was struck at a depth of 1,690 ft. in the top beds of the Dakota.

In March, 1915, two wells near the discovery well reported oil. The Heron-Elder well, on the western limb of the anticline, reached the top of the Dakota at 2,746 ft. Oil came into the well at 2,774 ft. and rose about 2,000 ft. The oil is dark in color and probably heavier than that from the discovery well. About a mile south and near the crest of the anticline the Western Pacific well reached the top of the Dakota at 2,150 ft. and report gives about 300 ft. of oil in the well accompanied by a strong gas pressure.

Location of Oil Discoveries

The locations of these wells may be found on Fig. 3. The townships are numbered from south to north and the ranges west from the initial meridian.

Moose Mountain well, Tp. 23, R. V, west of 5th meridian, about the center of the northern boundary of the township.

Discovery well, southwest corner of Tp. 20, R. II, west of 5th meridian.

Heron-Elder well, southeast corner of Tp. 20, R. III, west of 5th meridian.

Western Pacific well, northwest corner of Tp. 19, R. II, west of 5th meridian.

Petroleum as Fuel under Boilers and in Furnaces for Heating, Melting, and Heat Treatment of Metals

BY W. N. BEST,* F. R. S. A., NEW YORK, N. Y.

(San Francisco Meeting, September, 1915)

INTRODUCTION

CRUDE oil attracted attention because of its excellence as a fuel for openhearth furnaces; for making crucible steel and brass; for melting copper, lead, tin, zinc, nickel, silver, malleable iron, gray iron; and for the production of steam in all sizes and types of boilers. For heating furnaces, mold-drying ovens, core ovens, ore roasters, calciners, hot-air furnaces, sand drying, asphalt road work, etc., its superiority over other fuels is due to the fact that the heat is at all times under perfect control, so that a constant temperature may be attained and maintained at the will of the operator. This means a maximum amount and uniform quality of work.

Crude oil is of two kinds, paraffine base and asphaltum base, and varies from 11 to 46 gravity Baumé. The light-gravity oil does not require heating before use; but oil of less than 20 gravity Baumé should be heated to just below its vaporizing point, in order to reduce its viscosity and to obtain the highest efficiency of combustion. Heavy California oil vaporizes at 230° F., Mexican crude oil at from 175° to 210° F., and Beaumont, Texas, oil at 142° F. Fuel oil vaporizes at 130° F.

Scientific Installation Essential

The burning of oil is an art based on science, and the "rule of thumb" should never be employed, especially in heating heavy crude oil. Thermometers should be used on both suction and supply pipes so that the fireman or smith will at all times know the temperature of the fuel. The indicators on such thermometers should be large enough to be readily seen by the operator, night or day.

It is necessary to have an unvarying pressure of steam or air through the burner, to atomize the fuel. Also, the air used to support combustion should be delivered through a nozzle under the burner at a constant pressure. The oil pressure must not fluctuate; under no circumstances should it vary more than 1 lb. The oil pump, which should be brass-lined, with the aid of the pulsometer and superheater, will insure a constant

* Engineer in Caloric.

circulation of fuel to the various burners, maintaining an unvarying pressure on the supply pipes. All oil pipes should be of adequate size and carefully laid.

A non-carbonizing burner of modern construction, that will externally atomize the fuel and distribute the heat over the entire width and length of the furnace or firebox, should be used. For a furnace, the use of only one burner giving a flat flame covering the entire hearth is always pref-



FIG. 1.—HIGH-PRESSURE OIL BURNER MOUNTED FOR MARINE OR STATIONARY BOILERS BURNING OIL OR TAR EXCLUSIVELY AS FUEL.

erable. If steam is used through the burner to atomize the oil, it should be taken from the top or dome of the boiler or superheater so that it will be as dry as possible. The atomizer orifice in the burner should be as small as practicable to provide for the minimum and maximum capacity of the burner.

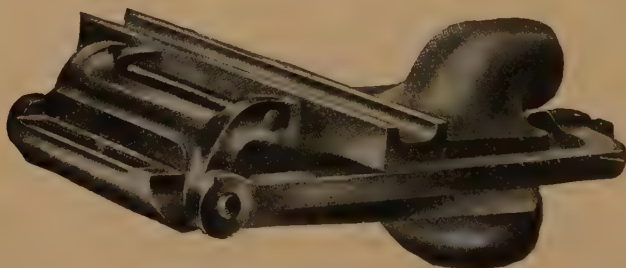


FIG. 2.—LOCOMOTIVE BURNER.

Combustion Chamber

The function of a combustion chamber is to provide space in which the atomized fuel may unite with sufficient air for combustion before it reaches the furnace proper. In the more modern furnace practice only a sufficient amount of steam or air is forced through the burner to atomize and distribute the oil, while the remainder (ordinarily about $1\frac{9}{20}$) of the air necessary for combustion is delivered at from 2 to 3 oz. pressure

through a nozzle, often located below the burner, and provided with a sliding gate, so that the quantity of air entering the combustion chamber is readily controlled by the operator. It is necessary that the flame made by the burner shall fit the combustion chamber perfectly, so that the volume air delivered into the chamber will pass through the flame. When this rule is observed, the melter cannot oxidize the metal, and injurious effects from sulphur in the fuel are prevented. The proportions of the chamber must be adequate to the size of the furnace and the temperature maintained therein.

BOILER SERVICE

The use of coal involves the handling of coal and ashes, the employment of additional firemen, etc., all of which is avoided by the use of oil. When, in view of this saving, the price of oil compares favorably with that of coal, the former is highly preferable on other grounds also. It is clean, simple and cheap in operation; and since it requires accurate control, it utilizes and trains the judgment of the fireman, and thus elevates the character of the labor employed.

In the equipment of locomotive, marine, and stationary boilers, the first consideration is the protection of the metal constituting the elements of the boiler. This can be effectively accomplished at a small cost.

Locomotive Service

As soon as a locomotive is changed from coal to oil fuel (which can be done at a very small cost), the train-tonnage of the engine is increased 15 per cent., because the locomotive can easily carry the steam pressure at all times at just below its "popping-off" point. This, of course, cannot be done while using coal as fuel.

For locomotive service the most modern practice is "the duplex oil system," which employs two burners, a small and a large one. The former, used as the engine leaves the roundhouse and operated continuously thereafter, serves as a pilot light, as well as to keep just sufficient heat in the firebox to maintain the temperature and the steam required when the locomotive is standing still. It keeps the steam at just below the "popping-off" point when only the air pump is running, and no other work is being done. The large burner, ordinarily placed above the smaller one, is operated when the locomotive is at work. By this system the life of the boiler is increased, and the handling of the locomotive becomes much simpler. Gravity oil feed is ordinarily used in locomotive service. Air pressure should not be used on the locomotive oil tank to aid in forcing the fuel to the burner; but in stationary or marine practice 10 lb. pressure should be maintained on the oil-supply pipe.

Marine Boiler Service

In marine service, oil, at the will of the fireman, readily increases the boiler capacity 50 per cent. or more. The fuel tanks on the vessel can be filled in a very few minutes, regardless of climatic conditions. There is no dust or dirt—a great advantage, since the dust from coal, settling in the state-rooms and around other parts of the vessel, is very annoying on shipboard. In an emergency, when the vessel must be run at maximum speed, liquid fuel quickly responds to the demand for extra work.

Power Plants

For stationary boilers, especially in electric-light plants, where there may be a battery of "standby" boilers, the use of oil permits the boiler capacity to be easily doubled in an incredibly short time, without injury to the elements of the boilers. One man can fire and water-tend twelve 300-hp. boilers.

COMMERCIAL FURNACES

Some years ago manufacturers often contracted for the complete installation of oil furnaces in their works, but they have found it better policy to have their own masons construct the furnaces, purchasing designs and oil burners, etc., from well-known engineers in this particular line. Under this system, greater care is exercised in the construction of the furnace, for the local mason knows that any errors or defects in workmanship will be brought to his attention, and he will be held responsible for them. Manufacturers have found, moreover, that their own masons can construct the furnaces for 50 per cent. less than any outside furnace manufacturer can afford to estimate. I do not here refer to openhearth furnaces. They require special brick, and workmen skilled in this particular construction.

Firebrick

The selection of firebrick should be carefully made and only the best quality used; for it requires as much time and labor to lay a bad brick as a good one. The brick should withstand the temperature required without dripping, disintegrating, or spalling. They must not be laid in a layer of fireclay, as is the practice with red brick, but simply dipped in a thin mixture of fireclay and water, before being placed in position in the walls or arch of the furnace. The fireclay should be purchased from the firm manufacturing the firebrick. If a different fireclay is used, it will not adhere so well to the brick, and furthermore, it will not fuse or bond with the surface of the latter.

In drop-forge, annealing, or heat-treating furnaces for soft steel,

carbon steel, or high-speed steel, the firebrick used should show no perceptible expansion or contraction when heated to the required temperature. Specifications for such a quality of brick are as follows:

	Per Cent.
Silica.....	56.15
Alumina.....	33.295
Peroxide of iron.....	0.59
Lime.....	0.17
Magnesia.....	0.115
Water and inorganic matter.....	9.68

Old Method vs. New Method

In annealing furnaces or furnaces for the heat treatment of steel, the use of pyrometers is essential, since the value of steel depends upon its heat treatment.

A few years ago, we often heard the expression, "bring the furnace to a cherry red for annealing," or, "for case hardening, etc., bring the furnace to a bright red or yellow." "Heat the steel to dark red, quench it and draw it to an indigo color or a peacock blue," etc., etc. Those expressions are almost forgotten now, and recording pyrometers are used instead. Without these recorders the output is not satisfactory to the operator, manufacturer, or user of the steel. Daily operations have now become a matter of official record. I know of one firm that used to import from a foreign country water in which to quench its steel, as the workmen considered the water found in the United States inferior for tempering steel. It is pleasant to add that this plant has now adopted modern ideas of shop practice, and is not quenching its steel in water.

Special Furnace Designs

Furnaces should be specially designed for each class of work. In the United States a manufacturer largely makes a certain specialty, and if he decides to introduce oil furnaces by reason of their generally reported merits—increased output, improved quality of product, etc.—he should take great care to get the size and form of furnace best adapted for heating his stock. For example, A has stock to be heated of the same size as his neighbor B; and yet A and B may require furnaces of different sizes, for the reason that A requires three times as long as B to bring his forgings to shape and size. B's furnace must have a charging space three times as large as A's, or else B's smith must wait for his stock to be heated instead of the metal being hot and ready for the man. Again, should A have a furnace of the same large size as B, his metal would be constantly waiting on the operator, which would result in considerable scaling of the charge, and a loss in metal and fuel. An engineer, going through the works of

various manufacturers, sees hundreds of such misfits, resulting in poor shop practice. We all agree that in the forging of iron and steel the metal should be removed from the furnace as soon as heated to the proper forging temperature, and at the same time the furnace must be of ample capacity to heat sufficient stock so that the smith will not have to wait for the metal to be heated.

Modern Smith-Shop Practice

In modern drop-forge shops and small heat-treating plants, the best practice is to have the furnaces so constructed as to be portable; that is, incased in a substantial shell and provided with lifting irons so that they can be easily handled by the shop crane. These furnaces should rest on two or more concrete columns, so that the charging space will be at a convenient height for the operator.

When the refractory lining has been burned out, the furnace is removed by the cranes to the mason's room, in which it is relined by a shop mason, while another furnace is substituted. In this way the hammers or tools do not remain idle, and the operator loses no time. These changes are made during the night at the smallest possible expense, a stock of relined furnaces being kept in the supply room ready for use.

The mason is provided with a workroom and blue prints, which enable him to make the particular form of refractory construction required for the type of furnace to be relined.

For drop-forge furnaces the use of magnesite bottoms is becoming very popular, because magnesite has no affinity for the hot metal, while clay brick or sand often adheres to the forging, which is very annoying to the smith.

Heat Treatment of Steel

As already remarked, the value of steel depends wholly upon the heat treatment which it receives. To obtain the desired results, it is essential to establish and maintain an even temperature throughout the entire length and width of the furnace. For the heat treatment of carbon steel, which requires an indirect-fired furnace, this can only be done by means of graduated heat ports. Only one burner should be used, the heat therefrom passing from the fire chamber into the charging space of the furnace through graduated heat ports substantially as shown in the accompanying Fig. 3. The size and location of these heat ports is an engineering problem requiring careful consideration; for if they are not scientifically and accurately proportioned the incoming air used for the atomization of the fuel or to support combustion will cause an excessive heat at the end of the furnace opposite the burner.

For high-speed tool steel a direct-fired furnace is necessary; and the

more modern types have a preheating chamber above the charging space. The waste gases from the lower chamber passing up into the preheating chamber, slowly preheat the charge before it is passed into the furnace proper, thus preventing the too sudden expansion of the metal. (See Fig. 4.)

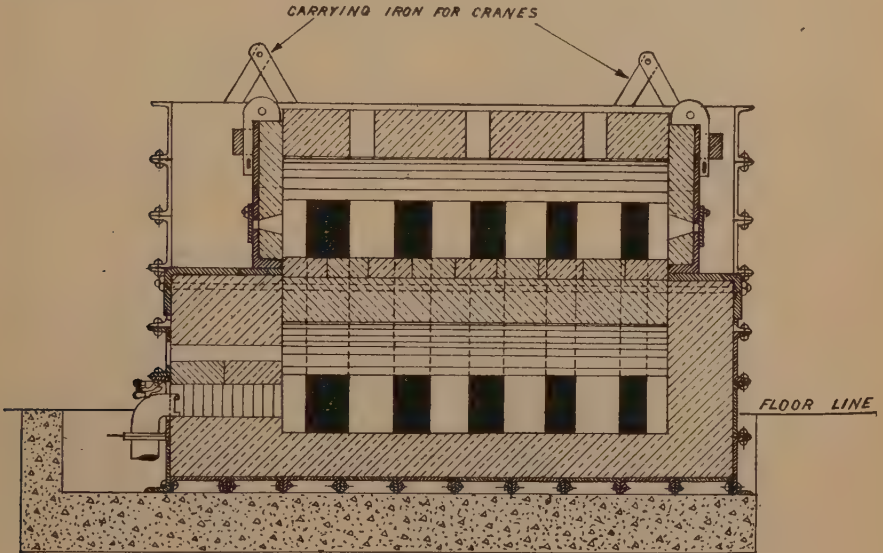


FIG. 3.—INDIRECT-FIRED FURNACE.

Brass Furnaces

The most modern practice is to charge the brass gates, old castings, etc., into a 2-ton or 4-ton melting furnace. From this furnace the metal is poured into ingots. These are tagged with the number of the heat and carefully analyzed by the metallurgist. When required these ingots are charged into a crucible furnace together with the required amount of copper, tin, or other alloy needed to meet the specifications of the purchaser.

A few years ago direct-fired furnaces were used in many brass foundries; but this method has been discarded as it resulted in an excessive loss in metal, while with oil-fired crucible furnaces the loss does not exceed that attending the use of coke- or coal-fired crucible furnaces.

Plate Heating

A localized heat is detrimental in a plate-heating furnace such as is used in boiler shops. Oil is an ideal fuel for this class of work since it insures an absolutely even heat throughout the entire charging space of the furnace, as well as a larger output than any other fuel.

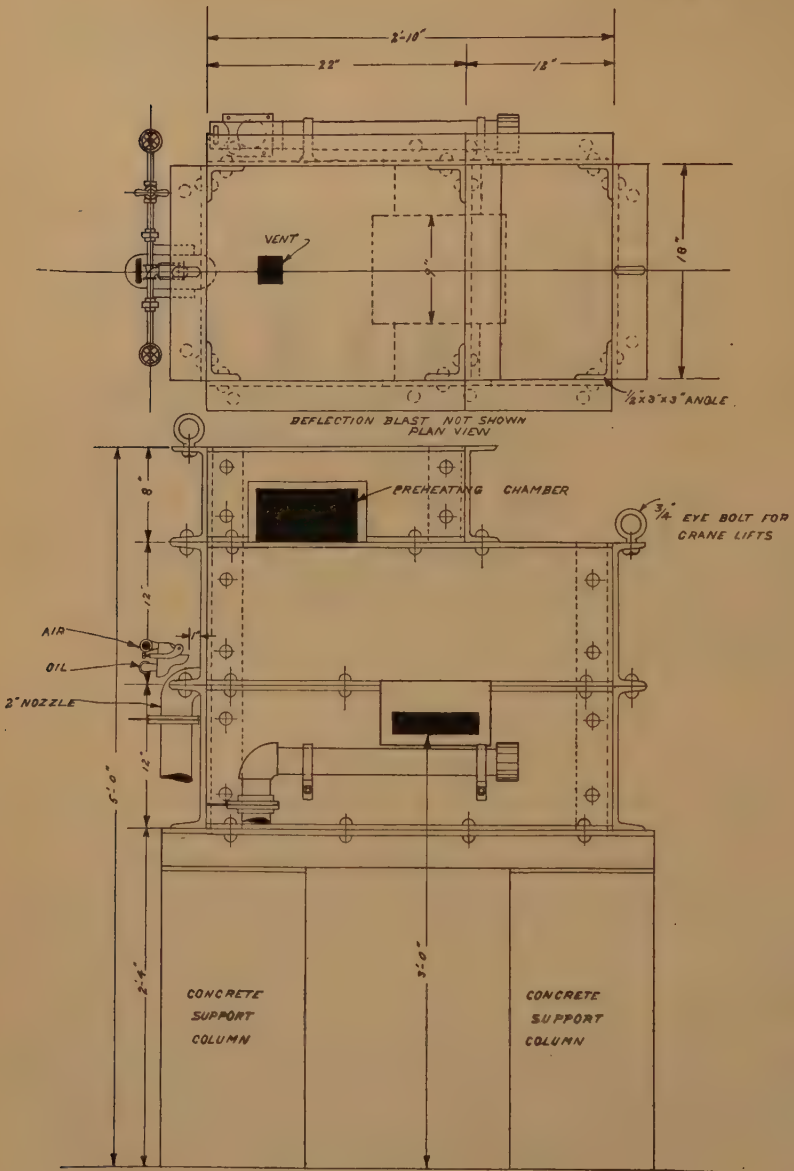


FIG. 4.—HIGH-SPEED TOOL-STEEL FURNACE.

Welding Furnaces

With coal- or coke-fired furnaces for this class of work, the pipe must be turned continuously; whereas, with an oil-fired furnace of proper construction, a flange can be welded on a 20-in. pipe without turning it; for the flame from the oil can be directed so that it will pass around the pipe and flange, producing a welding heat without moving the pipe. This furnace may also be used for vanstoning, etc. (See Fig. 5.)

Since it makes such a clean fire, oil is the best fuel for a billet-heating or scrap-welding furnace; and such a furnace formerly coal fired, when properly remodeled for the use of oil, will show a much greater output. Should there be any sulphur in the liquid fuel, it will not have any detri-

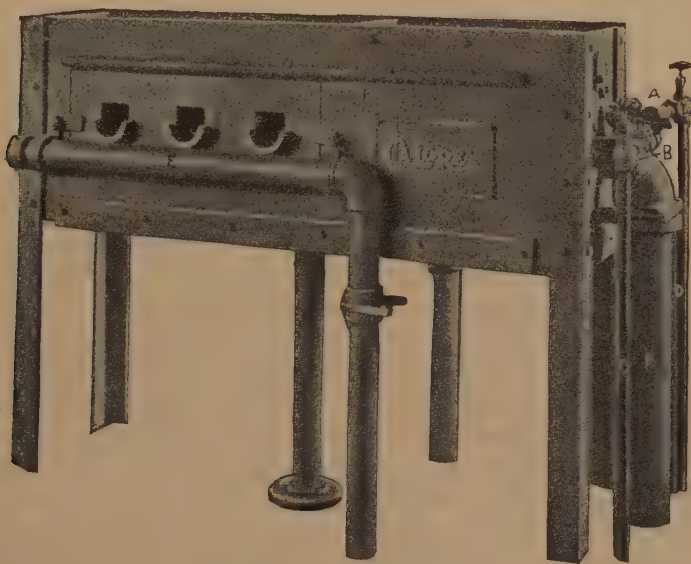


FIG. 5.—WELDING FURNACE.

mental effect upon the metal, if the combustion chamber is of the proper form and proportions.

Many persons are under the impression that they can compare the relative values of coal and oil by calculating the calorific value of the two fuels. This, however, is not true. The only accurate way to test the two fuels is to compare them in various classes of service. For example, in welding safe ends on flues for locomotives or stationary boilers, with a modern flue-welding machine a blacksmith and two helpers can weld 16 flues per hour in a coal forge, whereas with an oil furnace the same force of men can weld 60 flues per hour. This is because three flues can be placed at a time in an oil furnace, and only one flue in a coal forge. More-

over, the smith must be constantly turning that one, to prevent the fire from burning a hole in it, and he must also put borax and sand upon the weld. The welds with modern oil-fired furnace are clean; it is not necessary to turn the flue, and it is unnecessary to place borax and sand upon the part being welded, the weld being perfect without it. In this class of service 58 gal. of oil are equivalent to 1 long ton of bituminous coal (2,240 lb.) with a calorific value of 14,200 B.t.u. per pound, while in boiler service the average is 147 gal. oil as equivalent to 1 ton of coal.

Foundry Practice

Some day not far in the future, oil will find its place in every gray-iron foundry in the United States. At present cupolas are used; but every one realizes that cast iron belongs to an unruly family and that it is materially affected by high or low temperatures. Again, the oxidation of the metal in coke-fired cupolas is excessive; but with an air furnace or other type of oil furnace, the oxidation is reduced to a minimum, the temperature of metal desired can be attained and maintained without variation from day to day regardless of climatic conditions, etc. I have prophesied that there is a great future for oil in this particular service, but like every other new thought, it must be developed and perfected.

For ladle heating, in steel foundries, gray-iron foundries, malleable iron foundries, brass foundries, etc., oil is far superior to all other fuels. The various metals must be heated to certain temperatures before being poured and one of the new theories advanced in the year 1915 is that the ladles should be heated to approximately the temperature at which the metal is poured. This sounds reasonable; for if it is essential that the metal be at a certain temperature when poured, it is also equally important that it be not chilled when poured into the ladles. Oil is the fuel whereby the ladles can be properly heated to the same temperature as the molten metal.

Drying Ovens, Etc.

In mold-drying and core-drying ovens, liquid fuel is a desideratum, for with it we can maintain an even distribution of heat throughout the entire length of the oven, no matter what may be its proportions. Of course this is impossible with coal or coke. As the operator has the fire under perfect control, the material can be dried as quickly or as slowly as desired. Oil produces a more penetrating heat than other fuels, and if so desired, considerably less time may be used in this process without injury to the charge. Again, no stack is required, which reduces the expense of both construction and operation. In this practice, heat-recording instruments should always be used, as a matter of precaution as well as economy. The records should be in the office of the superintendent every

morning. It is surprising how much fuel can be saved by the use of heat-recording instruments and a sharp supervision of the work of the operator as indicated by the daily records.

The use of hot-air furnaces has had remarkable growth during the past two years, because they are clean and admirably adapted for the distribution of heat; moreover, they can be located in a small building some distance from the house, office, or factory, if desired, or they can be installed in the cellar or basement or in any other portion of the building. There is no fuel superior to oil in this class of service.

There has been a greater development in furnace construction during the past four years than in all the world's previous history. It requires 2,009 cu. ft. of air to furnish the oxygen requisite for the combustion of 1 gal. of oil. Only about 20 per cent. of this air is oxygen, while the other 80 per cent. is nitrogen and other gases which unfortunately must be heated up to the same temperature as the furnace, but, for economy's sake, must be expelled as quickly as possible. It is therefore necessary that the furnace construction be such that the consumed and inert gases will be expelled at once. The combustion chamber should be so located and of such form and proportions as to insure perfect reverberation of the heat, while aiding in the quick expulsion of the gases. Only one burner should be used on any billet-heating furnace, no matter whether the charging space be 12 by 18 in. or 8 by 24 ft. in size. For a regenerative melting furnace two burners are necessary; but of course only one burner is in operation at any one time. In all other melting furnaces, whether the bath be 2 ft. square or 20 by 140 ft., whether the product requires an oxidizing or a reducing flame continuously or alternately, only one burner should be used. Additional burners make the operation of the furnace too complex. While the operator may strive to run them all alike, some will be giving an oxidizing and others a reducing flame, thereby impairing the efficiency of the furnace.

In the designing of furnaces the question is not how many burners can be used, but how few will do the work. A poorly constructed furnace will ruin the efficiency of the best oil burner ever made, while a poor burner will ruin the efficiency of the finest furnace ever designed. It is as impossible to cover a flat furnace bottom or charging space with a round flame as it is to fill a square hole with a round plug. To obtain the highest efficiency and strictest economy it is necessary to have a good furnace, scientifically designed and constructed, as well as a superior type of oil burner.

DISCUSSION

WILLIAM A. WILLIAMS, San Francisco, Cal.—I would like to ask whether Mr. Best ever had any experience with an oil burner that used neither air nor steam?

W. N. BEST, New York, N. Y.—Yes. This type is commonly known as the “mechanical spray burner” and uses oil heated under high pressure to above its vaporizing point. These burners are not as efficient as the atomizing type for the following reasons:

1. They produce a round flame which, as with any other type of burner, must pass over a flat surface. A round flame cannot fit a flat surface.

2. They do not evenly distribute the heat, resulting in localized heat.

3. The outlet through which the fuel passes is very small; so all gravities of oil, especially oil containing sand, cannot be burned.

- 4. Their construction is such that they must be cleaned frequently, as the residuum in the fuel clogs the burner.

5. The oil opening being small, the capacity of each burner is small, so that many burners must be used in a boiler or furnace. (The fewer burners there are, the more accurately they can be regulated.)

In marine boilers, these burners are largely used, because they do not require either steam or air for atomization. However, there are other methods by which this objection to other types can be eliminated, such as by the use of a rotary blower and steam turbine engine, the exhaust from the engine passing into the engine condenser, thus obviating loss of water.

Mechanical burners are rarely used in shops. I know of but one power plant using them.

With mechanical burners, the fuel must be maintained at a high pressure. On board naval vessels, some mechanical burners demand an oil pressure of from 100 to 600 lb. It is advisable to reduce the pressure as much as possible, as fractured pipes or fittings are particularly dangerous, for the heated oil escapes as a vapor. In such cases there is liability of an explosion caused by spontaneous combustion.

HENNEN JENNINGS, Washington, D. C.—Is crude petroleum required for use under boilers, or is it possible to extract certain constituents of petroleum and then use the residuum under boilers? Is it not, with the limited production in the country, necessary to make a full use of petroleum before using it as a finality in producing steam?

W. N. BEST.—I have always contended that any burner which will not atomize any gravity of topped oil, crude oil of any gravity, fuel oil, oil tar, or coal tar is not worthy of the name burner. We have boilers equipped using coal tar as fuel, others oil tar, others crude oil as low as 11° Bé. gravity (Mexican oil), while some plants use oil containing as high as 3 or 4 per cent. sand; in fact all kinds of liquid fuel.

DAVID T. DAY, Washington, D. C.—What is the size of this 3 or 4 per cent. of sand, and what amount of dirt can go through your process?

W. N. BEST.—In the Mexican fields the oil is stored in large reservoirs or lakes and in pumping it up into tanks you will often find that it contains some sand of average size; therefore, a burner must have the oil opening of sufficient size to allow this sand to pass out readily. The oil opening should be independent of the steam or air atomizing opening so that the force of the steam or air will not cause the sand or other residuum in the fuel to cut away the metal of the burner. For this reason the exterior atomizing type of burner is always preferable. The average mechanical burner has an opening about $\frac{3}{64}$ -in. in diameter and any sand in the fuel must of necessity cut away the metal in the opening of the burner or sprayer.

In southern California, some oils contain as much as 2 per cent. sand, so fine that it cannot be separated from the oil by the use of strainers. With some types of burners the mouth is cut away by this fine sand. In such burners it is necessary to have removable parts in order to save the body of the burner. It is always advisable to have an abundant supply of repair parts on hand for an internal atomizing burner.

We often burn coal tar containing as much as 30 per cent. free carbon; the burner must be so constructed that the free carbon can readily pass through it.

MARK L. REQUA, San Francisco, Cal.—Do you know of any burner adapted to domestic use which can be installed at comparatively moderate cost? Burners are being used in apartment houses, hotels, and residences, but they cost so much that it is not feasible to put them into small houses. There is a field for a small burner if installed cheaply that would supplant the use of coal. Do you know of any burner of that type?

W. N. BEST.—There are a number of air carbureting burners which require no mechanical means for the atomization of the oil. The oil must be volatile (a distillate oil) so that it will readily burn on a hot plate.

In this hot plate are a number of openings to provide the requisite quantity of oxygen for burning the oil. And around the plate there is a refractory lining, the heat therefrom aiding combustion.

These burners can be installed very cheaply, the only difficulty being that they cannot be installed in some States in dwellings, etc., because of the fact that they ordinarily use gravity feed. A large number using gravity feed are in use in southern California, and in the Eastern States they are being used in small boilers for heating greenhouses and the like.

MARK L. REQUA.—There is a hot plate which uses hot water and oil.

W. N. BEST.—The trouble with this type of burner is that the oil used is not sufficiently volatile. A very volatile oil, a distillate, is required. Again, oil and water do not mix well and the quantity of water in this type of equipment must be accurately controlled. If anything goes wrong with the oil supply, too much water is fed and the apparatus is flooded.

A. F. L. BELL.—I would like to answer Mr. Jennings' question, as I do not think it was fully covered. Mr. Jennings wanted to know whether all petroleum should not be topped before using it. Practically all light petroleum in California is topped, as about one-half the fuel oil consumed in California is residuum oil; the other half is crude oil, too heavy for refining purposes.

M. L. REQUA.—I think Mr. Jennings has in mind the idea that to burn any oil before it is subjected to some refining process is not an economical way.

As far as the product resulting is concerned, the final residuum is as good for fuel purposes as crude oil, and it has practically the same number of heat units, or, in fact, a few more. And it does remove a certain economic waste which exists in burning the crude oil before it has been treated. The sooner the oil operators realize that they are wasting resources which should be utilized, the better it will be, I think, for all concerned.

Gasoline from "Synthetic" Crude Oil.

Continuation of the discussion of the paper of WALTER O. SNELLING, presented at the New York and San Francisco meetings, February and September, 1915, respectively, and printed in *Bulletin* No. 100, April, 1915, pp. 695 to 704 and *Trans.*, li, pp. 657 to 671.

A. C. McLAUGHLIN, San Francisco, Cal.—It seems to me that Mr. Snelling's work, if it is what it appears to be, is one of the most important contributions to petroleum technology that has been brought out in the last 25 years. If his end products are what they seem to be, he has made a discovery, which, in my judgment, will revolutionize the petroleum industry.

As to industrially carrying out these processes I do not think there is the slightest difficulty, for I have had some work recently where crude oils were handled successfully at a temperature of 750° F. and pressure of 750 lb. per square inch.

WILLIAM A. WILLIAMS, San Francisco, Cal.—What metal did you use, Mr. McLaughlin?

A. C. McLAUGHLIN.—Ordinary seamless tubing, extra strong.

WILLIAM A. WILLIAMS.—Tests were made in Dr. Rittman's experimental plant at Pittsburgh, upon samples of residuum furnished by Mr. Bell, the residuum being the remainder of the crude oil after the lighter fractions had been removed.

After the material had been run through Dr. Rittman's plant, yields of from 43 to 58 per cent. of material, boiling under 150° C. were obtained. The distillate had a peculiar odor, due to the presence of benzene, toluene, and other unsaturated hydrocarbons.

W. N. BEST, New York, N. Y.—What is the difference between the Rittman and the Snelling process or system?

WILLIAM A. WILLIAMS.—The fundamental difference is that reactions take place under relatively higher temperatures and pressures and with none of the oil present in its original or liquid state, while the reactions secured by Mr. Snelling and other investigators have been from the liquids in the presence of some vapors.

Under such conditions as Mr. Snelling operates, the temperatures and pressures used are limited and the flexibility in the control of these variables is extremely limited when compared with the Rittman process.

In making gasoline, Dr. Rittman has found favorable temperatures around 500° C. and pressures around 150 lb. to the square inch. Dr. Rittman has recovered from a 300° distillate as high as 85 per cent. of the original product which contains more than 30 per cent. which will boil under 150° C.

A. C. McLAUGHLIN.—What strikes me as most important, in connection with this Snelling discovery, is that it is so different from all cracking processes worked on during the past 30 years. The process of cracking is not new, and my experience as a refiner taught me that the cracking of oil was analogous to what is accomplished with the microscope, *i.e.*, there is a gain in magnification but a loss in definition; what is gained in the stills is lost in the agitators. Compounds are formed which must be taken out with sulphuric acid.

Now, it appears that Mr. Snelling has produced low-boiling, saturated compounds. If he has done that, he has done something never achieved before, and I regard it as a big advance.

M. L. REQUA, San Francisco, Cal.—Has anybody made any estimate of the cost?

A. F. L. BELL, San Francisco, Cal.—In connection with the tests of which Mr. Williams speaks, I do not know that any estimates have been made of what a practical working plant would cost, since the process is in the experimental stage.

The samples Mr. Williams returned to us were tested at our refinery laboratory and re-run as we do our ordinary tops. The highest distillates that came off, or that we could get off, were about 50° Bé., but they had the same boiling points and corresponded closely, by the Engler test, to the 60° Bé. gasoline refined from crude oils and marketed here. But when the samples were treated with sulphuric acid in the same manner as our ordinary distillates, there was a marked reaction; the sulphuric acid turned the distillates black and threw down an excessive sludge, making it impossible to clean them with sulphuric acid. Clarifying with lye and by filtration was only partially successful. We obtained products of a beautiful red color, and some with green hues. But our samples were so small (only 4 oz.) that we could not determine what process would make them a commercial product. I do not think, however, that they can be treated in the same manner as the ordinary distillates of petroleum. They do not have any of the repugnant odors that the ordinary distillates of petroleum have, when heated to the same temperatures in the ordinary still under atmospheric pressure.

WALTER STADLER, San Francisco, Cal.—In reading over the literature which has come to my notice regarding the Snelling system, it has been observed that most of the oils that have been subjected to the

treatment and on which detailed results have been published are those from the eastern United States and from Oklahoma. The crude oils from these regions, as I understand them, are usually composed of saturated hydrocarbons of the paraffine series, or a mixture of such saturated hydrocarbons with unsaturated hydrocarbons of the olefine series.

According to the results noted by Mr. Snelling, the process must virtually consist of converting higher-boiling heavy hydrocarbons of both the paraffine and olefine series into lighter low-boiling hydrocarbons of the paraffine series, since by repeated treatment a crude oil resembling the original oil is each time produced from which low-boiling members without color or odor can be distilled.

In California we are naturally interested in the California oils. As yet, detailed results from California oils subjected to the Snelling process have not been noted by me in the literature put forward. In this regard I may have overlooked something. The bulk of our California crude is entirely different than the Eastern and Oklahoma oils in that it is composed of unsaturated hydrocarbons to a degree of unsaturation frequently greater than in the olefine series, of naphthenes, some of the aromatic hydrocarbons, and minor amounts of the paraffine hydrocarbons.

If from the complicated California oils repeated treatment of the heavy residues will each time yield results comparable with those obtained from the Oklahoma and Eastern oils and give lighter products without color or odor, it will mean considerable for the technology of petroleum. It will mean, at least in part, a virtual reversal of the old cracking process. It will mean a process in which the more stable lower-boiling bodies are produced without the use of catalyzers. Results with California oils will therefore be of considerable interest.

A. F. L. BELL.—Our California oils are of asphaltic base, whereas most of the Eastern oils are of paraffine base, but I am under the impression that California oils can be treated as well in the Snelling process as the Eastern oils. I simply judge that from the treatment of the California oils in the Rittman process. Mr. Williams informed me that he was able to obtain a higher percentage of gasoline from the California oils than from the Eastern oils.

WALTER STADLER.—Were not those samples treated with sulphuric acid, and did you not get large quantities of tar which would indicate cracking?

A. F. L. BELL.—Yes.

DAVID T. DAY.—I would like to mention two points: First, concerning the specific gravity of the distillates, it seems odd to an Eastern man

to obtain gasoline, by means of the Rittman process, the specific gravity of which is very high. The explanation is simple: When these oils crack and as the temperature goes up, more and more of the aromatic series are produced. If the temperature is kept low, so that the oil is just cracked, a different specific gravity is obtained. The specific gravity of all these cracked products is high as compared with neutral gasoline.

Second, the matter of the purification of these cracked distillates is one of great importance. Such a large proportion of unsaturated bodies is present in the cracked gasolines that it would be out of the question to purify them by the ordinary methods employing sulphuric acid or alkali. Within the last 30 days I have been developing a process for refining the Rittman gasolines which is absolutely satisfactory, simple and cheap. So far I have not used any sulphuric acid. The refined product may not be as sweet as ordinary gasoline, but it is plenty sweet enough.

Underground Mining Systems of Ray Consolidated Copper Co.

BY LESTER A. BLACKNER, M. E., RAY, ARIZ.

(San Francisco Meeting, September, 1915)

THE PROPERTY AND LOCATION

THE Ray Consolidated Copper Co.'s mining property is located on Mineral Creek, 6 miles north of Kelvin, at Ray, Pinal County, Ariz. (Fig. 1).

The mining claims now owned by the company consist of 126 lode claims, comprising 2,144.9 acres, containing at the beginning of operations 82,904,368 tons of 2.19 per cent. copper ore.

GENERAL CHARACTERISTICS OF THE OREBODY

The main orebody is a disseminated deposit in schist and porphyry formed by secondary enrichment. Its existence was proved by churn drilling, and it is at present one of the largest proved copper deposits in the world. The orebody itself covers 205 acres with an average thickness of 121 ft., and occupies a definite belt which has a northwest direction. Its length is approximately 5,000 ft. The width varies, being about 2,000 ft. at the west end and 2,500 ft. at the east, narrowing down irregularly from both ends to a few feet in the center. The thickness of the body varies greatly along the line of lode, ranging from a few feet up to 470 ft. thick.

The ore horizon is not constant, but varies, following in a broad sense the topography. The body in general dips slightly to the northeast, and is broken up by numerous small faults and fractures. The ore-bearing formations, consisting of mineralized schist and mineralized granite porphyry, stand fairly well and offer no difficulty in mining operations.

The bulk of the ore is chalcocite disseminated in schist. In places cupriferous pyrite is closely associated with the chalcocite. The mineralized granite-porphyry formations are contiguous to the ore-bearing schist.

Overlying the orebody is an oxidized zone of leached iron-stained schist, averaging 252 ft. in thickness, termed "capping." The line of

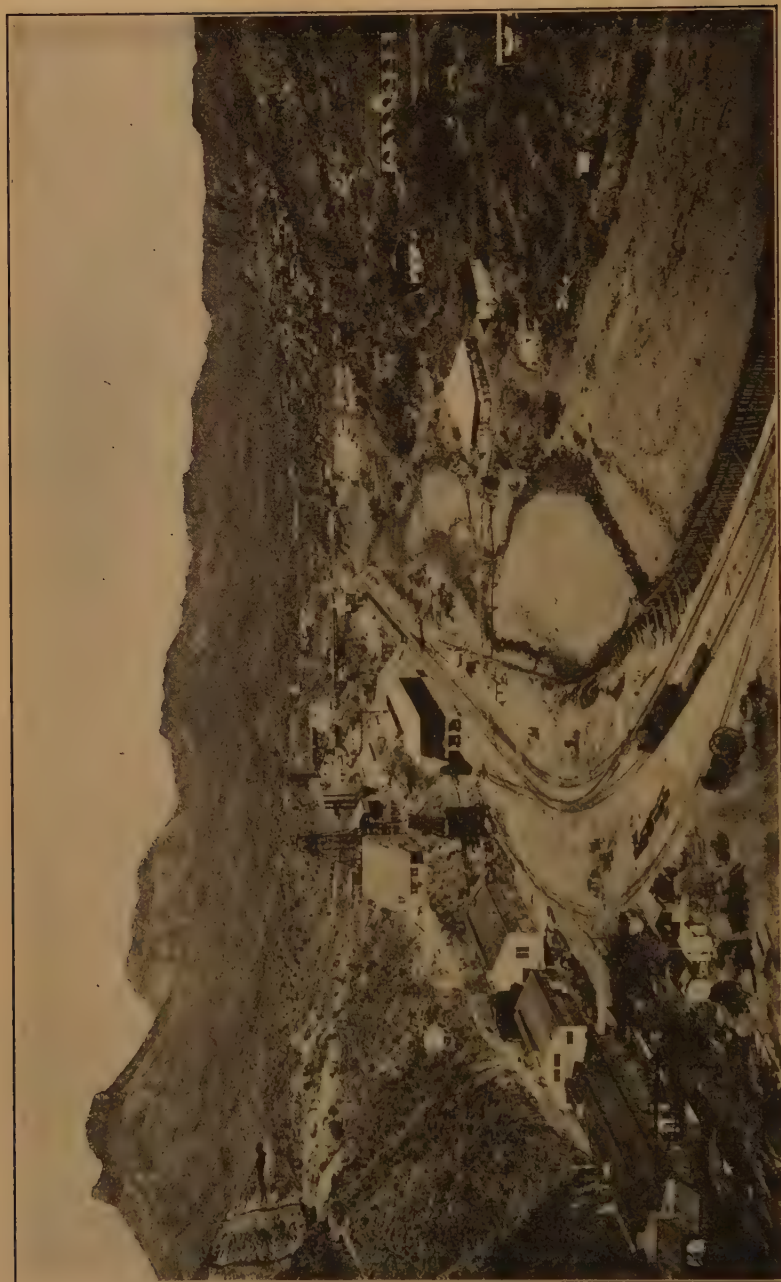


FIG. 1.—MINES OF RAY CONSOLIDATED COPPER CO., RAY, ARIZ. TEAPOT MOUNTAIN IN THE DISTANCE.

demarcation between the ore and capping is easily discerned, owing to the difference in color. In a few places along this line small amounts of the carbonate and silicate of copper occur. Within the orebody a little native copper has been found, and at the base of the oxidized zone small areas contain cuprite.

The foot wall of the ore in many cases is terminated by a diabase intrusion, and in other cases fades off into *protore*, that is, material which by the continuation of the process of natural enrichment might be converted into ore. The ore along the diabase contact is of higher grade than the main orebody.

DETERMINATION OF TONNAGE AND VALUE

In arriving at the tonnage and copper contents of the orebody every precaution was taken to secure accuracy. The developing was done with churn drills and underground drifting and raising. A complete survey of the property was made and the ground covered by a system of north-and-south and east-and-west coördinate lines 200 ft. apart. Churn-drill holes were put down as nearly as possible at the corners of the 200-ft. squares thus blocked out. The sampling was done very carefully. For every 5 ft. of drilling each hole was thoroughly cleaned and the sludge passed through a specially devised sampler. The samples were assayed locally and composites sent to outside laboratories for confirmation. Each drill hole was given a number, and in the office a sheet was kept showing in minute detail the geology, the men doing the work, time required, and assays for every 5 ft. of drilling. Records were also kept to determine the final cost per foot of drilling for each hole.

In determining the tons and assay value of the ore in any block the procedure was as follows: Each 200-ft. block was calculated separately. A survey of each block was made and the horizontal distances between holes determined. With these distances determined the actual area of the rectangle formed by the four holes was computed. After figuring the average for all the assays of that which could be called ore in a hole and multiplying it by the total number of feet of ore, the foot-percentage for the hole was obtained. Proceeding in a like manner the foot-percentage of each hole at the four corners of a block was found. Then after adding all the ore footages (ore in each hole) of the four holes together and dividing this total into the total of all the foot-percentages, the assay value in copper of the total tons of ore in the block was obtained; next by finding the average thickness of the ore in the block (thickness of the ore in each of the four holes added together and divided by four) and multiplying it by the area of the block and dividing by the number of cubic feet to a ton ($12\frac{1}{2}$) the total tons in a block was arrived at. Example:

A block having holes 251-249-267-263 for the four corners; area of the rectangle formed by these holes is 84,404 sq. ft.

Number of the Hole	Thickness of Ore, Feet	Average Assay of the Hole, Copper, Per Cent.	Foot-Per Cent.
251	90	1.77	159.30
249	345	1.86	641.70
267	100	2.65	265.00
263	80	2.62	209.60
Totals and averages	615	2.07	1,275.60

Average thickness of the ore in the block = $\frac{615}{4} = 153.75$

$\frac{84,404 \times 153.75}{12.5} = 1,038,169.20$ tons of 2.07 per cent. copper ore

By this method the tonnage and the assay value of each and every block of ore were determined. Then by adding the values for all the separate blocks, the total tonnage and the average assay for the entire orebody were obtained. As a check, the total tonnage was also figured by planimentering cross-sections of the orebody at 200-ft. intervals and computing the number of tons contained therein.

Of the 353 holes drilled to date, only 238 are inside the orebody and considered in calculating the ore tonnage; they represent 106,971 ft. of drilling. It is gratifying to note that all the underground development and prospecting thus far have checked with accuracy the churn-drill results both as to assays and tonnage.

SYSTEMS OF MINING

Owing to the heavy overburden and the low grade of the ore, caving systems have been devised and adopted which consist in weakening a block of ore by means of a series of shrinkage stopes or "ore-filled rooms." Then after undermining and shattering the remaining pillars the ore is drawn systematically, the capping crushing and settling gradually over it. Throughout all the work at Ray two systems have been used: The sub-level or motor-haulage system, employed in thick uniform blocks of ore; and the hand-tramming system, used in the shallower portions. From the accompanying sketches it will be seen that the method of carrying up the stopes and the manway arrangement are practically the same in both systems, the essential difference being in the method of handling the ore.

The same conditions as exist at Ray had been previously encountered at the Boston mine of the Utah Copper Co., Bingham, Utah, and the systems of mining initiated at that place were subsequently chosen for Ray. In explaining the "Ray systems" now in use it is necessary to go over important changes and steps at Bingham prior to their final adoption at Ray.

SUB-LEVEL OR MOTOR-HAULAGE SYSTEM

At Boston Mine, Utah Copper Co.

During the experimental stages of the system, stopes 20 ft. wide were carried up on 50-ft. centers, leaving pillars 30 ft. wide between stopes, as shown in Figs. 2A to 2D. In all the early operations at Ray as well as at Bingham the main difficulty was with manway connections or entrances to stopes and methods of undermining pillars. Raises at 100-ft. intervals were run up in the centers of alternate pillars and at points every 50 ft. up these raises small crosscuts were run out, from the raise to the edges of the stope on either side, to provide manway connections or entrances to stopes. Men entering a stope passed up through the raises to the first crosscut above the broken muck in the stope. However, this means of entry did not prove satisfactory, as the pillars usually faulted or sloughed before the stope reached capping, cutting off the means of entry.

The next method was to run "manway drifts" parallel to the center line of stopes down the centers of the 30-ft. pillars on sub-levels 50 ft. apart, as shown in Fig. 2C, with crosscuts or entrances to stopes 100 ft. apart. In addition to this means of entry, "pole roads" 100 ft. apart were carried up along one side of a stope, these being located opposite the crosscuts on the sub-level. The pole roads consisted of slabs (that is, segments of round poles) 6 to 8 ft. long, placed horizontally, one directly over the other, 2 to 4 in. apart and in front of a groove or niche in the side wall of a stope. These poles kept the muck from entering the manway. The groove was cut into the wall deep enough to allow a man to pass through with comfort. In order to enter a stope by means of the pole road the men climbed up ladderways in the center of the pillar from the motor level to sub-level 2, then through a small crosscut to the pole road, then up the pole road to whatever level the stope may have reached (see Fig. 2D); or else they could walk down the manway drifts on either sub-level 3, 4, or 5, then through a crosscut to the pole road, and up through it to the stope. Thus it can be seen that in theory the entries were convenient and plentiful. However, such an expensive network of manways did not work out in practice, as the pillar drifts, owing to crushing and faulting of the pillars, in places had to be timbered, thus becoming expensive to maintain. The pole roads also gave trouble, as too much muck had to be blasted into them, in making the niche, to be handled conveniently on sub-level 2, and at times the side walls around the niches broke wide, and expensive blocking and timbering were necessary to keep them open. At Ray, the pole roads were replaced with manways made of frame cribbing, so that the latter difficulty was overcome.

UNDERGROUND MINING SYSTEMS

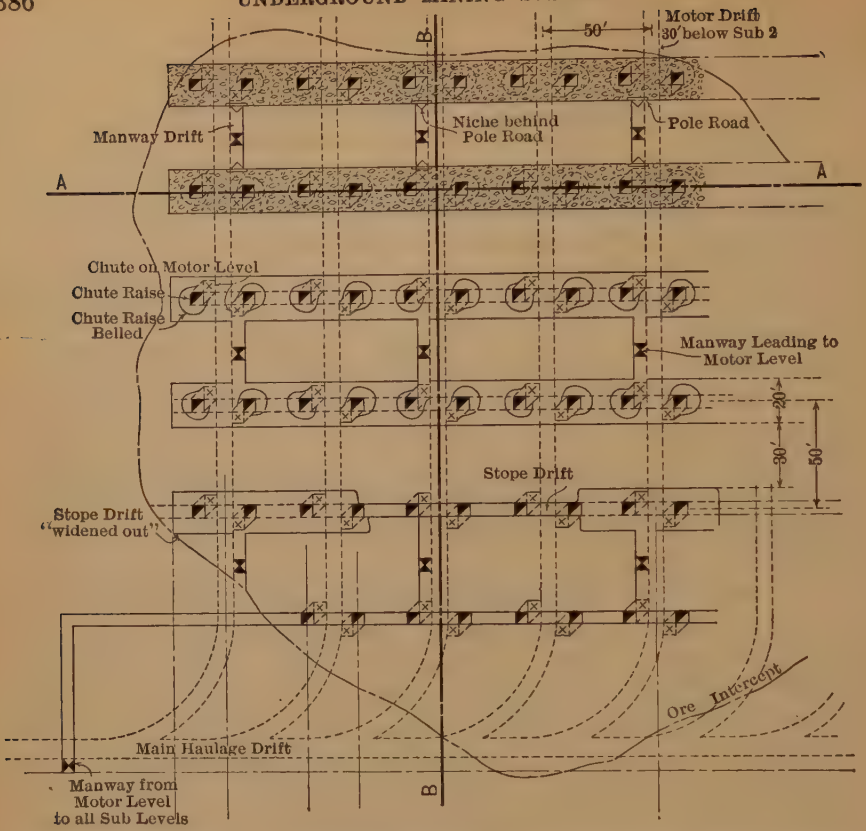


FIG. 2A.—SUB-LEVEL 2. MOTOR-HAULAGE DRIFTS ON MOTOR LEVEL DOTTED.

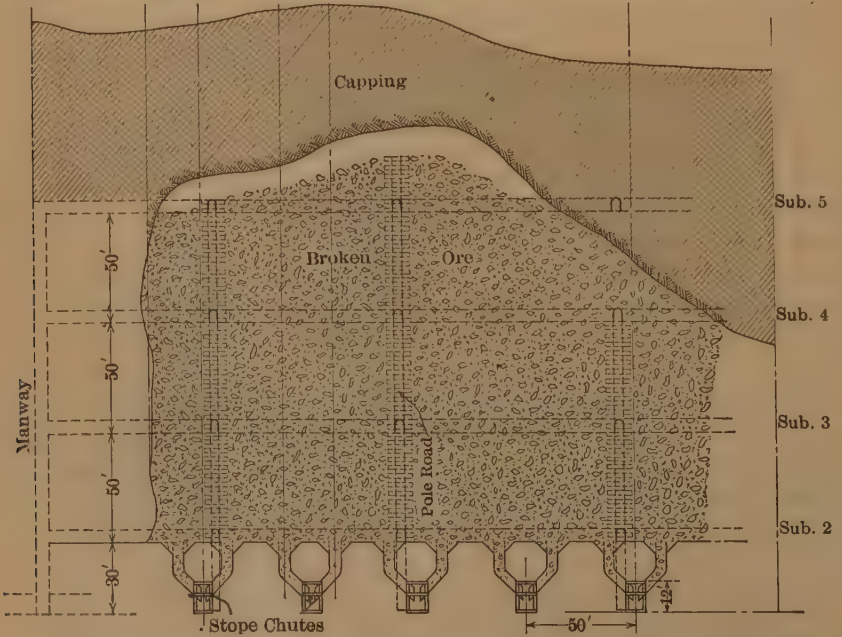


FIG. 2B.—SECTION A-A, SHOWING SUB-LEVELS AND POSITIONS OF POLE ROAD.

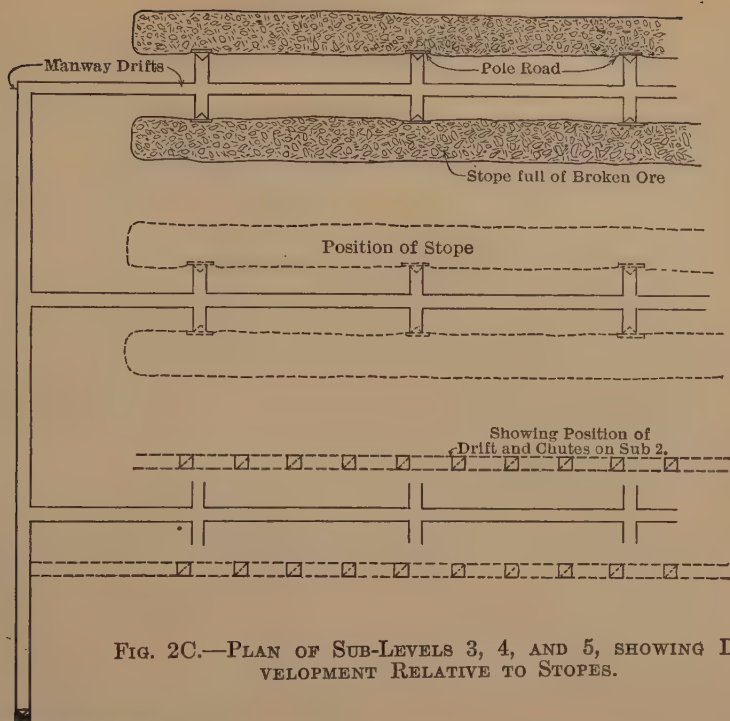


FIG. 2C.—PLAN OF SUB-LEVELS 3, 4, AND 5, SHOWING DEVELOPMENT RELATIVE TO STOPES.

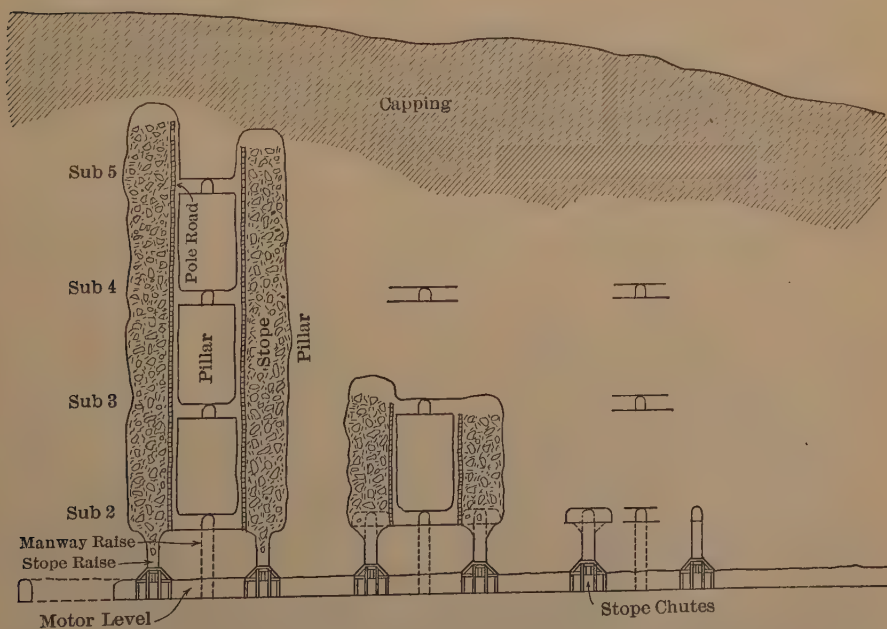


FIG. 2D.—CROSS-SECTION ALONG B-B, FIG. 2A.

FIGS. 2A TO 2D.—SYSTEM OF MINING USED AT THE BOSTON MINE, UTAH COPPER CO., BINGHAM, UTAH, AND IN EARLY OPERATIONS BY RAY CONSOLIDATED, AND FROM WHICH WAS DEVELOPED THE "RAY SYSTEM" OF UNDERGROUND MINING WITH MOTOR HAULAGE.

At Ray Consolidated Mine: Early Method

At Ray, the 30-ft. pillars, as explained in the foregoing, in portions of the orebody, did not crush as was expected, and it was necessary to put up regular shrinkage stopes, termed "pillar stopes," 8 to 10 ft. in width, in the centers of the pillars. Then it was found that even the remaining pillars, 10 ft. in width, between the pillar stope and original stope, did not cave, and that an additional small stope had to be run up, perhaps 8 to 10 ft. From the above it can be seen that this method of blasting or undermining the 30-ft. pillars became expensive, awkward, and unsystematic, especially when it is remembered that manways had to be provided for the pillar stopes and all the above work done before a block of ground could be drawn as a reserve. In addition, the drawing-off chutes along the haulage drifts on the motor level (see Fig. 2A) were put in staggered fashion on 10-ft. centers with loading platform between them, and the raises from these chutes were run up vertically to the sides of the stope above instead of on an incline to the center. This was abandoned because the raises coming up on the sides of the stope made it difficult to draw the stopes evenly, thus hindering drilling operations in the stopes themselves.

In order to overcome these difficulties a decided change in the arrangement was adopted: All stopes to be 15 ft. wide and spaced 25-ft. centers instead of 50-ft. centers, leaving a pillar 10 ft. wide, which could easily be undermined and shattered; also to have but two sub-levels; the first sub-level 30 ft. above the motor level, and the second sub-level, manway or ventilation level, near the top of the orebody, or about 100 ft. above the first sub-level, and that the drifts on the second sub-level be run at right angles to the center line of the stopes instead of in the pillars or parallel to the center line of the stopes, as was the former practice, and that manways be provided by running up raises from the first sub-level to the second sub-level at intervals of 100 ft. along the stope.

At Ray Consolidated Mine: Present Method

To simplify the present system of mining with sub-level it is expedient to classify the work into different stages (Figs. 3A to 3D).

First Stage: Drifting on Motor Level.—The orebody covers such an area that it has been found more economical to mine it from two main shafts. There is, however, a third shaft at which the ore is high grade, and is being mined by the square-set method.

At shafts 1 and 2 the "Ray system" is being used and the mines are opened up by three motor-haulage levels. On each motor level a main drift (double track, 30-in. gage) with two compartments (each 7 ft. wide, 8 ft. high, timbered with 12 by 12 in. posts, 12 by 14 in. caps,

6 by 12 in. sills, and 10 by 12 in. collar braces 4 ft. 2 in. long so as to place posts on 5-ft. centers) runs out from the hoisting shaft for a short distance to provide storage and passageways for trains and then narrows down to a single-track drift which is extended out along one side of the orebody. From this main drift a system of parallel drifts are turned off on a 60-ft. radius curve at intervals of 50 ft. and these are extended entirely through the orebody to a "fringe drift" which follows the ore intercept. Somewhere along the main drift, or one of the parallel drifts, in a convenient place outside the orebody, a "permanent raise" is run up to the sub-levels. Such a raise is usually widened out and cribbed into two compartments (cribbing 6 by 10 in. by 9 ft. long, dapped 1½ in. deep and 6 in. back on the narrow sides of both ends and also in center for spreaders 6 by 10 in. by 5 ft. long. Spreaders are all dapped 1½ in. deep and 6 in. back on the narrow sides of both ends). One compartment is used as a manway (size 3 ft. 6 in. by 4 ft.), the other (size 4 ft. by 4 ft.) for hoisting supplies such as steel, pipe, hose, cribbing, powder, etc., for the stopes.

Second Stage: Chute Building on Motor Level.—Usually as the parallel motor drifts are driven they are timbered within the ore limits with 12 by 12 in. by 7 ft. 6 in. posts, dapped 1 in. deep on three sides at upper end to receive the cap and collar braces and 1 in. deep on drift side of bottom end to receive sill. Caps are 8 by 12 in., 9 ft. long, dapped 1 in. deep on all four sides at the ends and 11 in. back on three sides and 8 in. on the fourth or top side (on one of the 12-in. faces) to receive pony posts. The cap rests flatwise on the posts. Sills are 6 by 12 in. by 9 ft. long dapped 11 in. back and 1 in. deep on the upper side of both ends to receive posts. Collar braces are 8 by 12 in. by 4 ft. 2 in. long, framed 1 in. deep and 1 in. back on one 8-in. face at the ends to fit in with cap. The caps and collar braces are made of light timber to be subsequently protected with "pony sets" (Fig. 4).

Outside the ore intercept 12 by 14 in. timbers are used for caps and 10 by 12 in. for collar braces.

Within the ore limits below an area which is to be stoped, pony sets are erected on top of all drift sets and are made up of 8 by 12 in. by 5 ft. posts dapped 1 in. deep and 1 in. back on three sides at the ends; 12 by 14 in. caps 9 ft. long, framed 1 in. deep and 7 in. back on three sides of both ends to receive posts and collar braces, are used. Collar braces are 8 by 14 in. by 4 ft. 2 in. long, dapped 1 in. deep and 1 in. back on the drift side of both ends to fit in with the cap, thus spacing drift sets on 5-ft. centers.

In heavy ground a 6 by 12 in. filler block is placed under the cap, and supported in center of cap by 12 by 12 in. angle braces 5 ft. long over all, extending diagonally down to corners formed by pony posts and top of cap of drift set (Fig. 4).

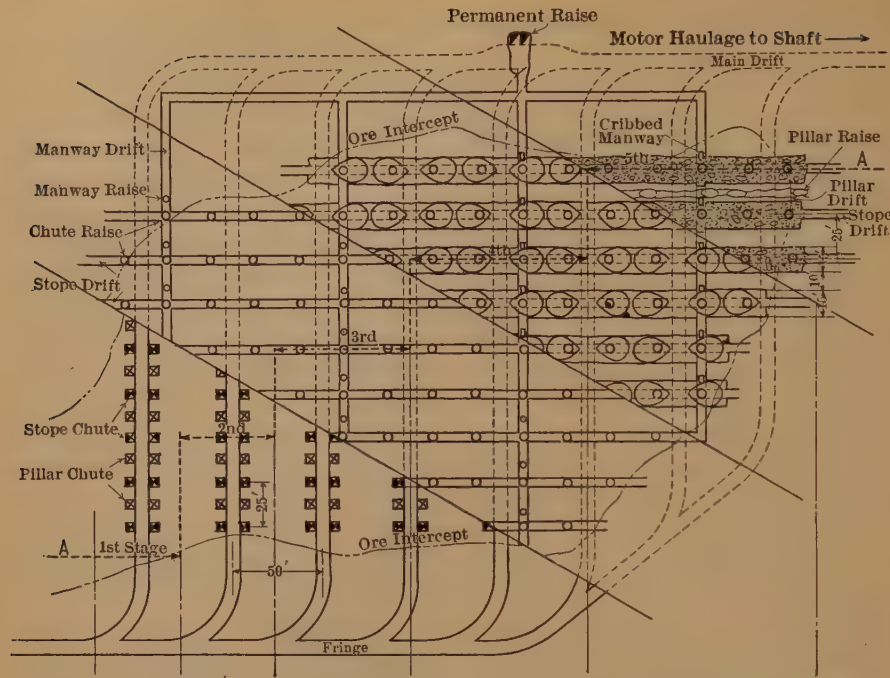


FIG. 3A.—PLAN SHOWING SIX STAGES IN DEVELOPMENT.

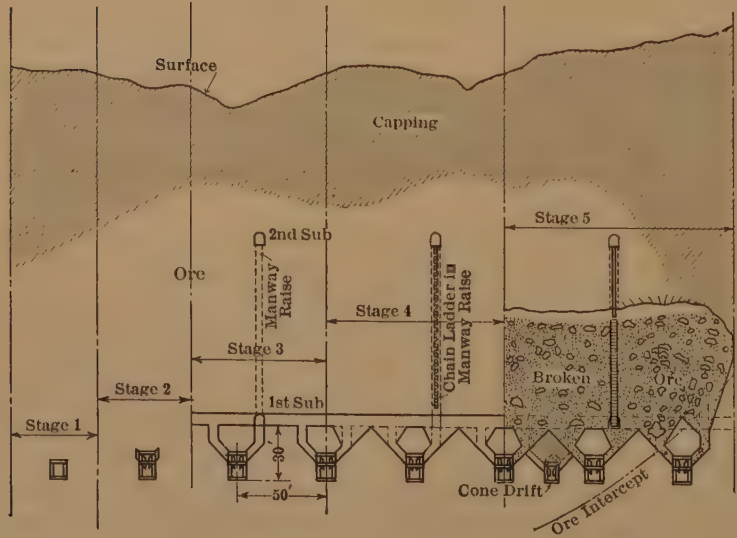


FIG. 3B.—LONGITUDINAL SECTION ALONG A-A, FIG. 3A.

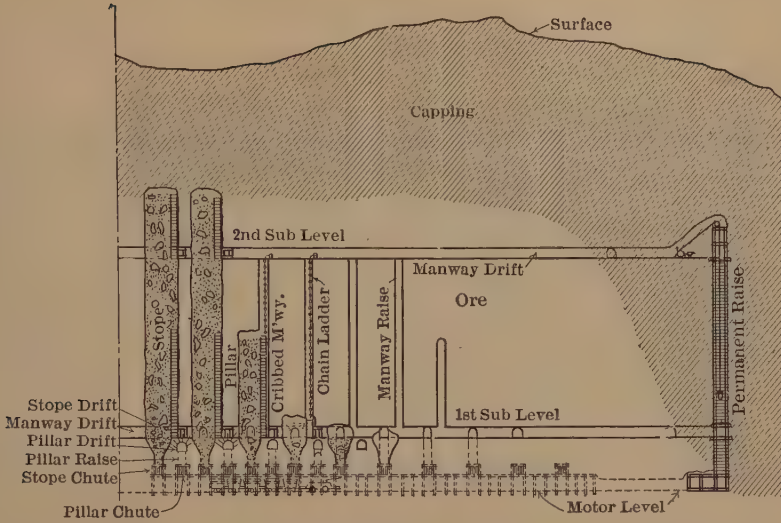
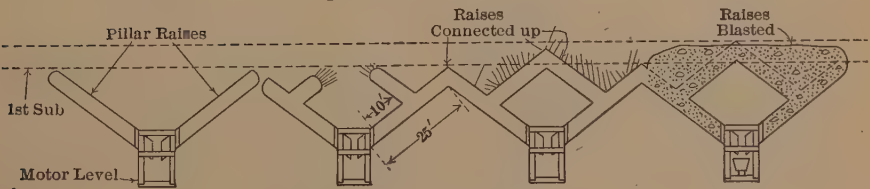
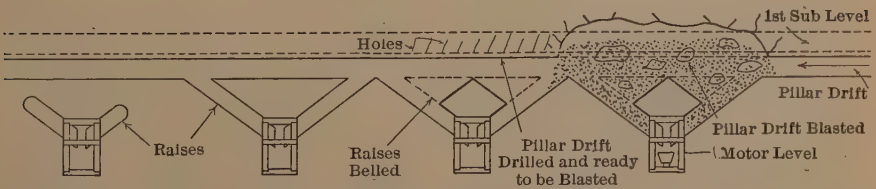


FIG. 3C.—CROSS-SECTION SHOWING SHRINKAGE STOPES AND PILLARS.



Method No. 1.—Pillar Raises.



Method No. 2.—Pillar Drifts.

FIG. 3D.—METHODS OF UNDERMINING PILLARS. STAGE 6.

FIGS. 3A TO 3D.—“RAY SYSTEM” OF MINING. SHRINKAGE STOPES AND PILLAR
CAVING WITH SUB-LEVEL AND MOTOR HAULAGE.

At intervals of 25 ft. along the drift, stope chutes are built opposite each other within a pony set so as to leave a 3-ft. clearance between them for muck to pass. Midway between, or $12\frac{1}{2}$ ft. from the stope chutes, pillar chutes are built. The chutes are made of 3 by 10 in. lumber, sides 3 ft. 6 in. long with 60° bevel on the two ends, and are nailed to pony posts. Bottoms are 3 ft. 6 in. long and nailed to the top of drift collar brace so as to have an incline of 30° for muck to run on. Chutes when finished are wide enough for a boulder 3 ft. in diameter to pass through (Fig. 5).

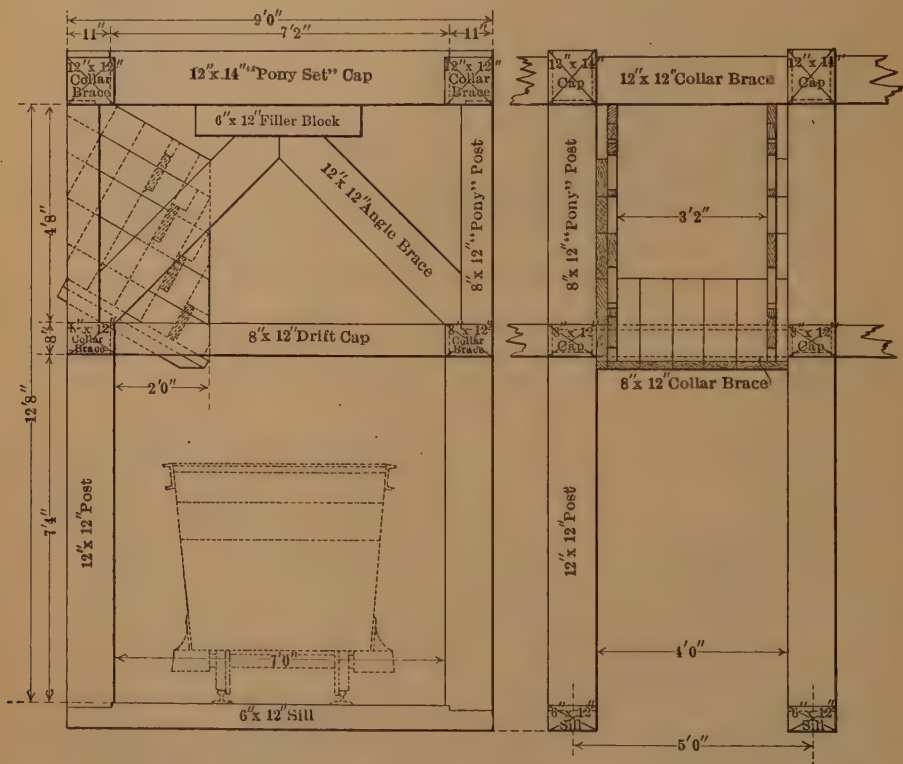


FIG. 4.—DRIFT TIMBERING FOR SUB-LEVEL, MOTOR-HAULAGE SYSTEM OF UNDERGROUND MINING.

Third Stage: Manway Drifts, Stope Drifts, and Chute Raises.—At the same time the motor drifts are being driven, small manway drifts, size 5 by 7 ft., are driven parallel to them at intervals of 100 ft. on the first sub-level 30 ft. above. These manway drifts are offset $12\frac{1}{2}$ ft. to one side of the motor drift so as to be directly over a chute raise, and out of them at right angles are run a series of parallel stope drifts, spaced every 25 ft. over the entire orebody. These stope drifts are placed so as to be directly above stope chutes on the motor level, so that when raises are run

up from the stope chutes they break into the stope drifts and become passages through which ore is drawn out into motor cars. The raises are usually 6 ft. in diameter and are run on an incline for 10 ft., then vertically until they strike the stope drift above.



FIG. 5.—MOTOR DRIFT, SUB-LEVEL SYSTEM. SHOWING PONY SET, CHUTES, ANGLE BRACES, AND A 5-TON CAR.

On a second sub-level, which is usually about 100 ft. above the first sub-level, manway drifts are driven parallel to and directly over those

driven on the first sub-level. Then later when connections are made they serve for ventilation and passages through which men and supplies enter and leave stopes.

Fourth Stage: Manway Raises, Belling Out Chute Raises, Widening Out Stope Drifts, Building Manway Sets.—Along the manway drifts on the first sub-level, at intervals of 25 ft. and $7\frac{1}{2}$ ft. from the center of each stope drift, manway raises are run up to manway drifts on the second sub-level. Chain ladders are put into these raises so that men may descend or ascend from one sub-level to the other, and later the raises serve



FIG. 6.—INSIDE A STOPE. SHOWING CRIBBED MANWAY BELOW MANWAY RAISE.

as means of access to stopes from the second sub-level. While the manway raises are being run, men with stopper machines go down into the chute raises to "bell" them out, and when finished the chutes have the appearance of funnels or inverted bells.

In starting a stope, men with stopper machines drill a line of holes all along and slanting into the sides of the stope drift. This line of holes when blasted, together with the work done to these drifts during "chute belling," widens them out to 15 ft. so that they are ready to start up as "active stopes."

As soon as the stopes are ready to start upward (Fig. 6), manway sets are erected in the manway drifts on the first sub-level. These sets

are built so that the 8 by 8 in. stringers on which the posts stand project out into the stopes, and a 3 by 3 ft. opening, made by cribbing (manway timbers) built up on these stringers, lies directly under a manway raise and serves as an entrance into an active stope from the first sub-level. The manway sets are made of 8 by 8 in. timbers, and the manway cribbing of 4 by 6 in. by 3 ft. 8 in. long, dapped 4 in. back and $\frac{3}{4}$ in. deep on the narrow side of both ends.

Fifth Stage: Mining Stopes to Capping.—In mining a stope (blasting down the back), two lines of holes, opposite each other, are drilled all along the back on both sides of the stope. The one line is drilled close to the side wall, with the holes slightly toeing into it; the other line, usually placed 4 ft. away from the side wall, is drilled so that the holes incline slightly toward the center of the stope. If the ground is very hard a third line is drilled still nearer the stope center. The machinemen working in the stopes use stoper machines and each man drills from 12 to 18 6-ft. holes per shift. These holes are usually loaded with four sticks of 30 or 40 per cent. powder; 6- and 7-ft. fuses are used, and each man blasts his own holes, going off shift. The oncoming shift draws off through the stope chutes into 5-ton motor cars on the motor level the excess ore, which is usually 33 per cent. of the total broken per round, thus giving the men plenty of headroom to re-drill the back while standing on broken ore.

The manways are built up with cribbing, when necessary, so as always to be open and above the muck. The machinemen in a stope receive air for their machines through a $1\frac{3}{4}$ -in. supply hose dropped down from the second sub-level through each of the manway raises. When a stope has reached a place about midway between the first and second sub-levels, the men naturally descend into the stope from the second sub-level through the manway raises instead of climbing up through the cribbed manways from the first sub-level. The cribbed manways become therefore a needless expense and are covered over the top with 2-in. lagging and left behind.

In hard ground the stopes are carried up 15 to 20 ft. wide, and in soft, "sloughing" ore 10 to 15 ft. wide.

Blacksmith shops are provided underground near the areas being stoped, so that men are quickly supplied with sharp steel and tools (Fig. 7).

Sixth Stage: Undermining Pillars.—After a block of ore has been weakened by a series of stopes the only step remaining in order for the entire block to crush and break is to undermine the pillars. Two methods are now in use:

Method 1: Pillar Raises.—Used in blocks of ore where the ground is hard and the stopes are carried up 15 to 20 ft. wide, leaving narrow pillars which are readily undermined. Starting with the pillar nearest the "fringe drift" on the motor level, raises are run out on flat inclines from

each of the chutes for the same pillar in the various drifts until they intersect each other (Fig. 3D, Method 1). Then out of these raises at a distance 10 to 12 ft. from the chutes, raises are run back so as to connect with each other directly over motor drifts. After the raises have been connected up all along a pillar they are widened out, lined with deep holes and blasted so as to undercut the entire pillar completely. Proceeding in a like manner with each consecutive pillar, the whole block is finally undermined.

Method 2: Pillar Drifts.—Used in blocks of ore when the ground is sloughing, badly faulted, and packs, so that for safety to the miners the original stopes had to be carried up narrow, 10 to 15 ft. wide.



FIG. 7.—UNDERGROUND BLACKSMITH SHOP. SHOWING STEEL SHARPENER TO THE RIGHT, AND HAND FORGE TO THE LEFT.

In the center of and all along a pillar between two stopes, at a height of 22 ft. above the sill of motor level, small drifts are driven with plugger machines (Fig. 3D, Method 2). Raises from the "pillar chutes" are run up on flat inclines until they intersect the drift. The drifts can be widened out as much as desired with stoper machines, to undercut the pillar. The raises are "belled," and the back of drift filled with deep holes and blasted so as to shatter the pillar completely.

The advantage of Method 2 is that a pillar of any width may be undercut and shattered; however, Method 1 is cheaper, and therefore is used where the pillars are narrow.

Seventh Stage: Reserve Drawing.—In most cases the orebody and capping, especially at No. 1 shaft, are badly shattered and broken up by small fracture seams so that when the pillars in any stoped area are undercut and the ore drawn, the capping directly over the outer edge of the area breaks in a nearly perpendicular plane to the surface.

In areas where the haulage drifts are to be abandoned as soon as all ore has been drawn, the procedure at present is to get the waste on a slight incline dipping toward the fringe drift. This is accomplished by drawing the chutes nearest the fringe more rapidly than those farther away, so that by the time the drifts take weight all ore will have been drawn and the occasional expense of retimbering, when all chutes in an area are drawn evenly, overcome.

An accurate account is kept of the ore drawn from every chute so that the ore remaining in each is always known. Only a few cars are drawn at a time, thus permitting the ore to settle gradually, with capping following after, thereby avoiding as much as possible the intermixing of the two.

For drawing chutes, empty cars are fed by the motorman through the fringe drift into the back ends of the parallel drifts, in which chutes are being drawn. These cars are then one by one taken ahead by a car pusher to the chute from which the ore is to be pulled. A chute blaster or loader stationed on a platform at the base of a pony set then lifts the boards or gates in the chute and allows the ore to run into the car. When a car is loaded a second car pusher shoves it down toward the main drift while his partner is spotting another empty for the chute blaster to load. All drifts are run on $\frac{1}{4}$ per cent. grade in favor of the load.

As soon as a train (usually 8 to 12 cars) has been loaded and pushed ahead, a motor comes through the main drift into the lateral; the motor helper couples the cars, and the motor then takes the train of cars to the shaft or tippie, where it is dumped, and the empty cars are returned to the back end of whichever drift may be in need of them.

The highest efficiency and best results are obtained when only two car pushers and one chute blaster (car loader) are used in each drift. Recently at the No. 1 shaft, six car pushers, three chute blasters, two machinemen, one mucker, and one chute repairer (timberman), working in four different drifts, loaded 400 motor cars, or an average of 150 tons per man, during a shift of 8 hr.

Machinemen drill and blast any large boulders which may lodge in a chute, also connect over raises preparatory for undermining pillars. The chute repairer replaces any broken chute boards, and makes other small repairs needed after blasting. The only men employed in a reserve-drawing section, other than those mentioned above, are timbermen on general repair work, a car checker, and a boss.

Eighth Stage: Cone Drifting.—After all chutes in a block have been drawn to capping, the next step is to recover the ore below the sub-level in the pillars between the motor drifts. This is done by first driving a small timbered drift parallel to the motor drifts in the centers of the pillars throughout the entire length of the block. Chutes are built opposite each other in every set along the drifts. A small stope extending along and directly over the drift is widened out and carried up to the sill of the first sub-level. Ore is drawn out into small cars (capacity 22 cu. ft.), which are then pushed by hand to a dumping chute, or winze, at the end of drift, where the ore is dumped to the next motor-haulage level below. When all chutes in this center cone drift have been drawn to capping, the small pillars still remaining on either side are taken out by slicing, so that eventually all ore above the sill of the motor-haulage level will have been extracted.

HAND-TRAMMING SYSTEM

This system differs from the sub-level method in that the stopes are started immediately above the drift sets and the ore is drawn out into cars of 22 cu. ft. capacity which are pushed by hand to dumping chutes.

At Boston Mine, Utah Copper Co.

Drawing-Off Chutes on One Side of Stope.—In the early experimental stages this method of stoping was very crude, as can be seen from Fig. 8A; on the tramming level a stope drift was run at right angles from a series of parallel drifts which had been spaced 30 ft. centers. The stope drift was widened out to 20 ft. by slabbing off the sides with "water-Leyner" drifters. At intervals of 60 ft. along the stope, pole roads were provided for men and supplies to enter and leave stope. These pole roads were made by building up slabs or round poles in front of niches in the side wall of the stope. At first the excess ore broken by the machinemen was drawn off below by men shoveling it into cars holding 28 cu. ft.; this was soon found to be expensive stoping, so chutes were built in the faces of each drift, and ore then drawn direct from stope through the chutes into cars.

The disadvantages were that only a small tonnage could be obtained, as only one stope could be worked at a time; the stope being drawn on only one side made it difficult for the machinemen to work; cars had to be loaded at the ends instead of at the side, and consequently were continually over-running, necessitating shoveling to clean track after each tram; low final extraction, as ore in far side of stope could not be recovered, and as can be seen from the present Ray method of starting stopes, the work of drifting and slabbing on the sill floor was a waste of time and

money, as the void made by this work was refilled with unrecoverable broken ore from the stope.

Drawing-Off Chutes on Both Sides of Stope.—In order to overcome some of these defects the scheme shown in Fig. 8B was adopted. Here the tramming drifts, 60 ft. apart, were run down the centers of the pillars between stopes, and at intervals of 30 ft. crosscuts were turned off from them at right angles to the stope on either side. This arrangement provided chutes 30 ft. apart on both sides and all along each stope, so that the ore in the stope could be drawn down more evenly and a higher final extraction obtained; this also made it possible to work two or more stopes at a time, giving a greater output. However, the stopes were started in much the same way as before in that a small drift was run down the center position of a stope, later slabbed out to 20 ft., or to whatever width it was decided to make the stope, and the stope started direct from the sill floor of the tramming level. The pole roads for manways were run up in much the same way as previously explained. The disadvantages of loading the cars at the end; the poor ventilation in stopes; difficulty with manways; waste of time and labor of slabbing on sill floor before a stope could be started, still continued. Also, by reason of the fact that the crosscuts leading from tramming drifts were directly opposite each other, too large an opening was necessary, so that pillars took weight, making it difficult to keep tramming drifts open. The arrangement of having the chutes along the sides of the stope made it necessary for the machinemen at times to work out in the center where muck was continually falling (sloughing) from the back, and also made it necessary while starting up a stope to shovel the ore along the center, over to the sides above a chute.

Drawing-Off Chutes in Center of Stope.—In the next block of ore, to further reduce difficulties, the tramming drifts were run down the centers of every alternate pillar, thus eliminating one-half of the development work, and instead of turning the crosscuts to the stopes out of them at positions opposite each other they were placed staggered fashion (see Fig. 8C) so that the smallest opening possible was obtained at every point along the tramming drift. The crosscuts were extended in each case beyond the center line of stopes and two chute sets erected in each to provide chutes directly under the stopes instead of at sides. In starting a stope, raises were run up on flat inclines out of each chute, until they intersected, thus forming "hog-backs" between each pair of crosscuts. This method of starting stopes was a decided improvement over the previous procedure, as it eliminated all the expensive drifting and slabbing on the sill floor under a stope, and made it possible to recover all ore broken in a stope. Manways were provided at first in much the same way as in early operations with the sub-level system as previously explained; i.e., raises were run up out of the tramming drifts at spaces

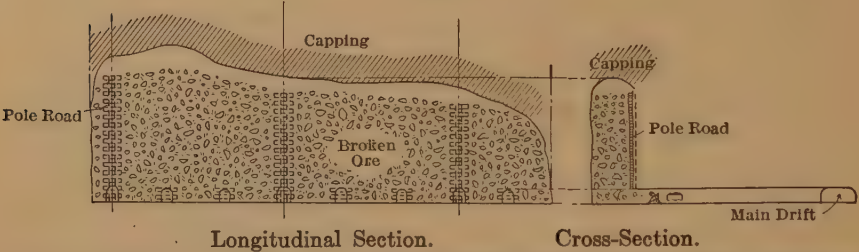
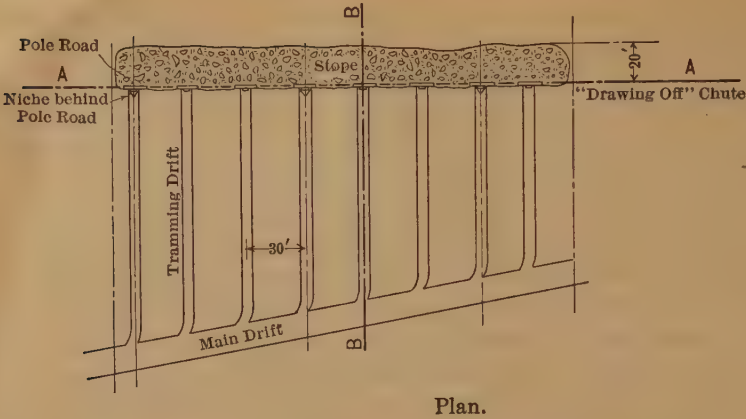
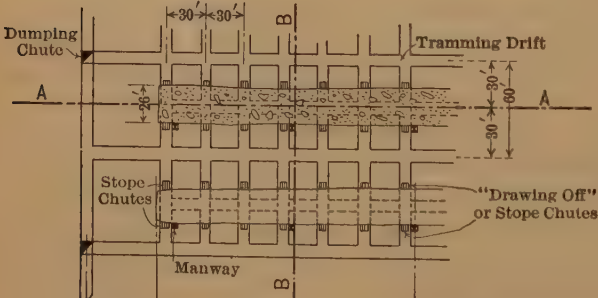


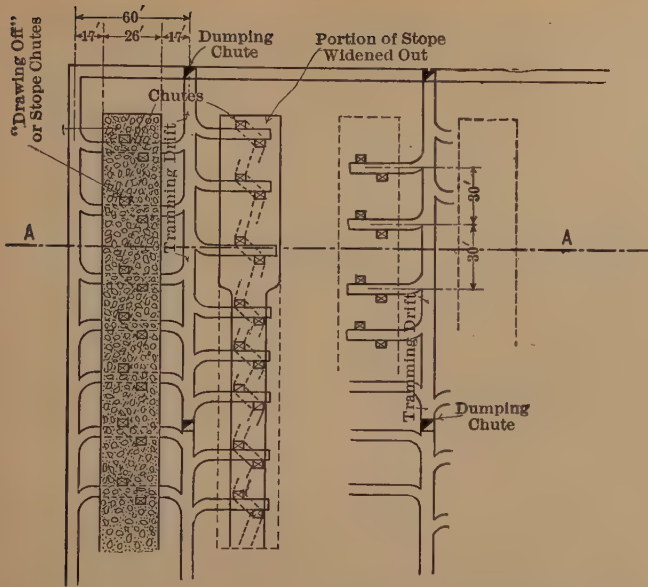
FIG. 8A.—SYSTEM OF MINING AT BOSTON MINE WITH DRAWING-OFF CHUTES ON ONE SIDE OF STOPE.



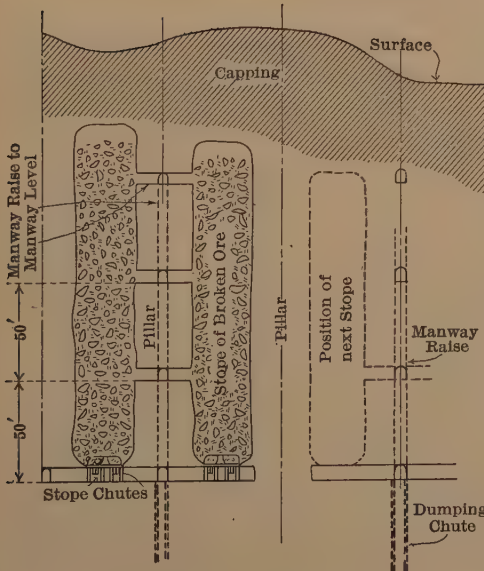
Plan, showing Arrangement of Stope Chutes and Dumping Chutes.



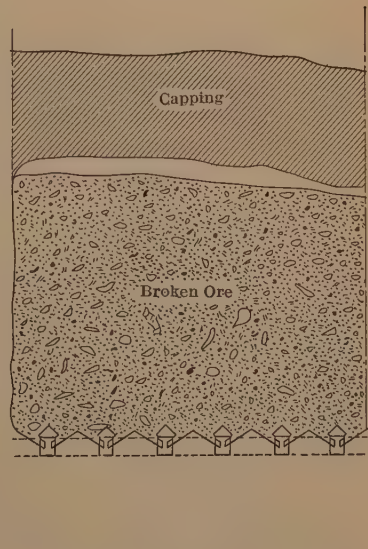
FIG. 8B.—SYSTEM OF MINING AT BOSTON MINE WITH DRAWING-OFF CHUTES ON BOTH SIDES OF STOPE AND TRAMMING DRIFTS IN EVERY PILLAR.



Plan of Tramming Level.



Cross-Section along A-A.



Longitudinal Section of Stope.

FIG. 8C.—SYSTEM OF MINING AT BOSTON MINE WITH DRAWING-OFF CHUTES IN CENTER OF STOPE AND TRAMMING DRIFTS IN ALTERNATE PILLARS.

60 ft. apart, and at distances of 50 ft. up these raises small crosscuts were driven over to the sides of stope to provide means for entering and leaving. These raises proved unsatisfactory and were abandoned, as the pillars became cracked and faulted, making them dangerous for the men to use, and necessitated driving drifts down the centers of alternate pillars between two stopes on the sub-levels, and out of them the crosscuts driven to hit the sides of stopes. These manway drifts proved satisfactory in the hard, uniform sections of ore, but when working in ground full of seams and faults they not only became impassable, due to breaking up of pillars, but the tramping drifts and crosscuts on the main level had to be held open by expensive timbering and retimbering. Such a condition obtaining at a time when active stoping was being done, it can easily be seen that heavy expense would naturally follow during reserve drawing of such blocks. In addition, the dumping chutes were poorly arranged for obtaining good results. Owing to the great distance to be traversed to dump their cars, the trammers unavoidably lost much time waiting their turns, consequently the output in tons per man was low as compared with results now being obtained at Ray. If reserve drawing had been attempted with such a system it would have been impossible to extract the ore in the pillars without driving additional drifts.

At Ray Consolidated Mine

Preparatory Work.—The system now in use is shown in Figs. 9A to 9D, and consists essentially as follows: All the tramping drifts are at right angles to the stopes, instead of parallel to them. Along two opposite sides of a block of ore fringe drifts are driven. Between these two drifts a system of parallel drifts on 25-ft. centers are driven (track 18-in. gage, 12-lb. rail). They are timbered with 12 by 12 in. posts 8 ft. long, dapped 1 in. deep and 1 in. back on three sides of the upper end to receive cap and collar braces and the same on one side of the bottom end to receive the sill. Caps are 12 by 12 in. by 6 ft. 10 in. overall, framed 1 in. deep and 11 in. back on three sides of both ends to fit on posts and receive collar braces. Collar braces are 10 by 12 in. by 4 ft. 2 in. overall, framed 1 in. deep and 1 in. back on one side of the two ends to fit in with cap, placing posts on 5-ft. centers. Sills are 6 by 12 in. by 6 ft. 10 in. overall, framed 1 in. deep and 11 in. back on the top side of both ends to receive posts. In sections where the ground is especially heavy a 6 by 12 in. by 2 ft. filler block is placed under center of cap, and supported by angle braces extending diagonally down on an angle of 45° to posts (Fig. 10). Angle braces are 10 by 12 in. by 3 ft. 6 in. long overall, beveled at each end, and are held in place by standards 6 by 12 in. by 5 ft. nailed to drift face of posts. Timbers 4 by 10 in. by 5 ft. long are generally used for top lagging, while 2 by 8 in. timbers, 3 ft. 10 in. long, are placed "louver"

fashion between posts as side lagging (Fig. 11). Along the sides of each of the drifts at points 25 ft. apart stope chutes are built. They are supported by 6 by 12 in. by 4 ft. chute sills placed horizontally between posts at a height of 4 ft. 11 in. above the sill. One edge of the chute sill is beveled so as to give a dip of 30° to the chutes, which project out into the drift just far enough to be over the side of a car. They are built of 2 by 8 in. timber. While the tramming drifts are being driven a permanent raise, providing the block of ore is thick enough to warrant such an expenditure, is run up in much the same way as described in the "Ray Sub-Level System" to a sub-level which is near the top of the orebody. From this raise on the sub-level a drift is run along one side of the block of ore, and is usually over, and parallel with, a fringe drift on the tramming level. Out of this drift, at intervals of 75 ft., manway or ventilation drifts are driven completely through the area to be stoped. These drifts are placed so as to be over tramming drifts, so that when manway raises are run up from manway chutes, which are built along the tramming drifts near the side of a stope to be, they break into the manway drifts and become, when chain ladders are put into them, openings through which men may pass up and down from one level to the other, and later serve as means of access to stopes from the sub-level.

Stoping.—In starting a stope, raises are run out on flat inclines direct from the stope chutes until they intersect, as was the final practice at Bingham. However, at Ray all drawing-off chutes (stope chutes) for a stope are along the same straight line in the centers of the stopes instead of staggered fashion. After the raises all along a stope have been connected they are widened out to give a stope 15 ft. wide, which is mined upward in the same manner as explained for the sub-level system.

Entries to the stopes from the tramming level are provided by building up a 3 by 3 ft. cribbed manway with the opening into the stope always above and clear of the broken ore. These manways are 75 ft. apart along the side of a stope and lie directly below the manway raises leading to the ventilation or manway level. By placing the cribbed manways directly under the manway raises very little ore falls into them, so they can, at the beginning of a shift, be quickly emptied, allowing men to enter the stopes without loss of time.

Undermining Pillars.—After all the stopes in a block have been finished the work of undermining pillars preparatory for reserve drawing is very simple, as the method is the same as for starting a stope. Pillar chutes, four in all to each pillar in each drift, are built in drift sets midway between each pair of stope chutes and each is so placed as to be directly opposite another. Raises are run out from these pillar chutes, and when all in the same pillar have been connected up, widened out and blasted, the remaining ore in the block, which amounts to 80 per cent. of the total, is ready to be drawn. Winzes, termed dumping chutes, to a

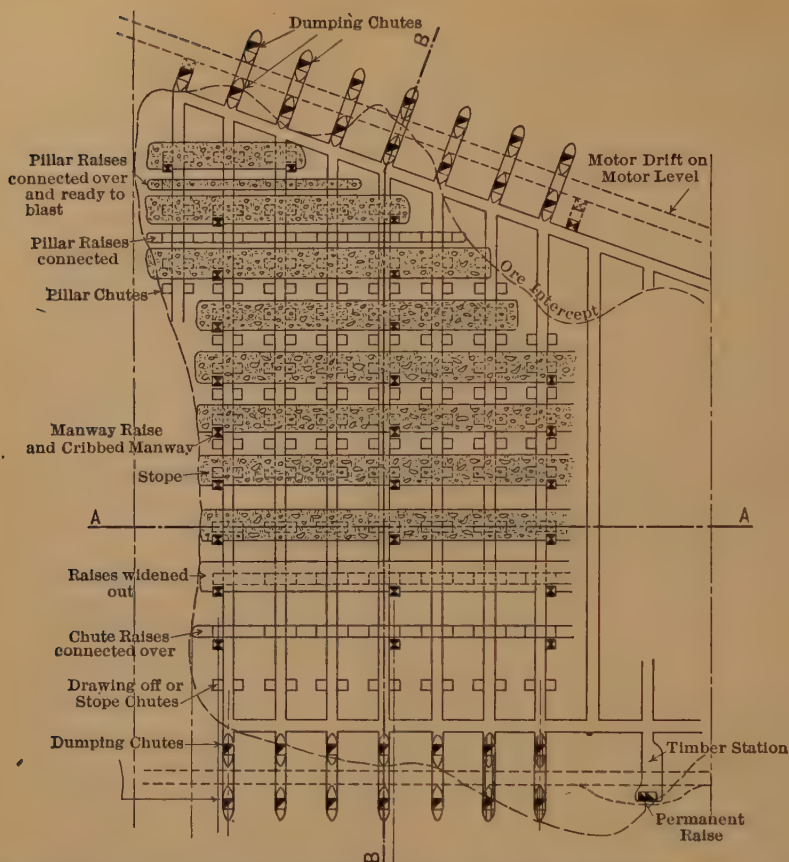


FIG. 9A.—PLAN OF HAND-TRAMMING LEVEL, SHOWING ARRANGEMENT OF STOPE MANWAYS, STOPE CHUTES, PILLAR CHUTES, AND DUMPING CHUTES.

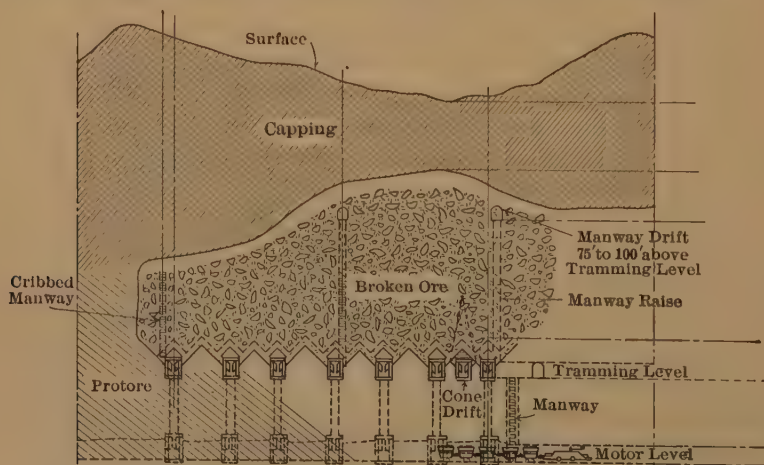


FIG. 9B.—LONGITUDINAL SECTION ALONG A-A, FIG. 9A.

motor level below are provided at the ends of each drift, and after the ore is dumped into these chutes by the trammers it is drawn out into the 5-ton motor cars and taken by electric haulage to the shaft.

Cone Drifting.—In actual practice it has been found impossible to recover all the 80 per cent., as some ore packs in the form of cones over the hogbacks between the trammings drifts; therefore, after all the chutes

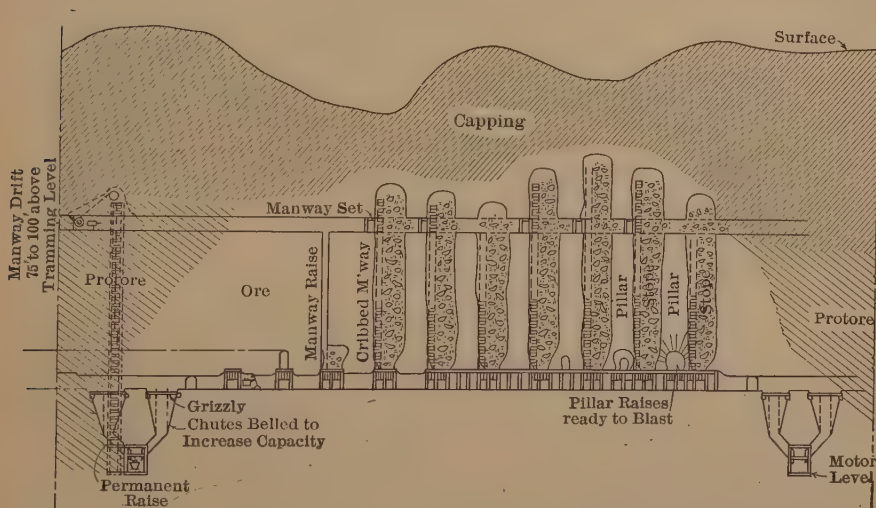


FIG. 9C.—CROSS-SECTION ALONG B-B, FIG. 9A, SHOWING SHRINKAGE STOPES AND PILLARS AND DUMPING CHUTES TO MOTOR LEVEL.



FIG. 9D.—METHOD OF UNDERMINING PILLARS.

FIGS. 9A TO 9D.—“RAY SYSTEM” OF UNDERGROUND MINING. SHRINKAGE STOPES AND PILLAR CAVING WITH HAND TRAMMING. DRAWING-OFF CHUTES IN CENTER OF STOPE AND TRAMMING DRIFTS AT RIGHT ANGLE TO STOPE.

in the original drifts have been drawn to capping, drifts termed cone drifts are driven down the centers of the 18-ft. pillars. These are timbered the same as the original drifts; chutes are built in every set and drawn, until capping appears. The additional ore remaining in the small pillars on either side of these cone drifts is recovered by gouging out along the sides of the cones.

Reserve Drawing.—In “reserve drawing” a block of ore (that is, extracting the 80 per cent. remaining after all active stoping has been completed) four factors must be kept constantly in mind: 1, recede

toward solid ground; 2, draw rapidly; 3, systematize the drawing; 4, repair broken timbers immediately.

1. Receding toward Solid Ground.—It has been found impracticable to draw large blocks of ore evenly over their entire area, owing to crushing of timbers, which necessitates high expenditures for timber repairs. However, in several blocks now completely finished good results were obtained by what is termed locally the "receding method." The two pillars farthest from the permanent raise were the first to be undermined. They, with the stopes adjacent, were the first to be drawn, and in order to have the waste or capping following down over the ore on an incline

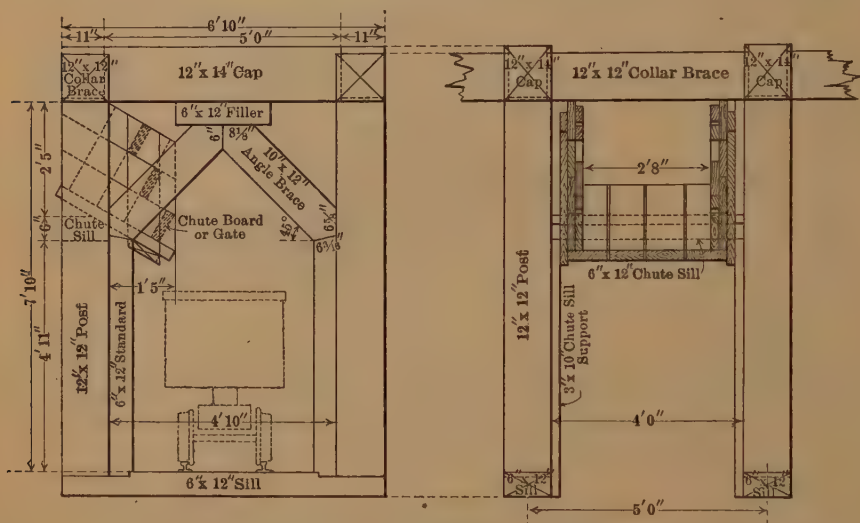


FIG. 10.—DRIFT TIMBERING, HAND-TRAMMING SYSTEM.

dipping away from the main block of stopes the chutes farthest from the permanent raise were drawn harder than those nearer. Not before, but by the time the first row of chutes had been drawn to capping, was the next pillar nearest the two already undermined, blasted, shattered, and made ready for drawing. Proceeding in such a manner no area more than 50 to 75 ft. in width was crushing and caving at any one time, and as fast as the line of chutes farthest away was drawn to capping it was abandoned, allowed to crush, and a new line provided by undermining the pillar next in order. The advantage of this process of receding was that any drift beneath the area being drawn could take weight and crush completely but still be recovered at a very small total cost.

2. Drawing Rapidly.—It is advisable to pull the ore out quickly once an area has been undermined, since by so doing in most cases all ore can be recovered without any timber repairs being necessary. The determination of the economical proportion of the non-producers, such as

timbermen, timber helpers, chute blasters, screenmen, car checkers, and bosses, in a section, to the actual producers, such as trammers and



FIG. 11.—TRAMMING DRIFT IN HEAVY GROUND. SHOWING CHUTES, ANGLE BRACES, STANDARDS, AND "LOUVER" LAGGING.

muckers, is essential in order to secure the lowest final cost per ton. In other words, as many trammers as can be used without their interfering

with each other should be employed in drawing any block of ground, and therefore it is usually advisable to provide a dumping chute at both ends of every drift. Pursuing this line of reasoning, it might seem advantageous also to have a dumping chute in the center of each drift, but the impracticability of such an arrangement has been demonstrated by the fact that the drifts are thereby weakened and crush, cutting off communication from end to end. Recently in a section with the dumping chutes arranged at the ends of the drifts eight trammers during a shift of 8 hr. produced 770 tons, or an average of 97 tons per trammer. Trammers are given a bonus for each car over a certain number, and are therefore at all times willing to put forth their best efforts.

3. Systematizing the Drawing.—Owing to the great number of chutes being drawn at a time a constant vigilance and record must be kept of each, so that not only the tons remaining, but the assay value, may be known at all times. With such information there is no reason why the total output in tons, assay value, and cost for producing same need fluctuate from day to day.

The duty of the car checker is to notify the boss at the beginning of each shift, or even the day before, just which chutes are ore and which are waste, so that he may know at all times which should be drawn, where to place his men for obtaining the most beneficial results and just when to undermine the next pillar. It can be seen that by such an arrangement the boss is responsible for the tons produced and the total cost for same, while the car checker must keep the assay value constant; and with such coöperation the best of results must naturally ensue.

4. Repairing Broken Timbers.—If the timbers in any drift crush it is essential to rush the repair work from both ends. A constant supervision of repairs needed must be enforced at all times in order that no ore may be left behind. In most cases drifts taking weight are detected immediately, and the work of drawing out the ore is rushed sufficiently to recover all ore before the drift becomes impassable, thus effecting a big saving for timber repairs. Most drifts crush very gradually, and none collapse suddenly.

HANDLING BROKEN ORE AND WASTE

On each motor level a 3-car tippie (see Fig. 12) placed directly over an ore pocket has been installed. The tippies are similar to those used in coal mining and are arranged to receive three 5-ton motor cars at a time. The 5-ton cars, in trains of 12 cars, are brought to the tippie by motors which weigh 10 tons each. When three cars have been spotted in the tippie by the motorman, the tippie operator turns on the electrical power and the tippie is rotated and the cars of ore turned completely over, allowing the ore to fall into an ore pocket. The pockets are

usually 10 by 28 by 25 ft. deep, holding from 250 to 300 tons. From a pocket the ore is fed through air-dropped gates (Fig. 13) to measuring pockets, size 4 by 4 by 12 ft., holding from 10 to 12 tons—a skip load. From the measuring pocket the ore goes through an air-lift gate to a 10-ton skip, size inside 4 ft. 8 in. by 5 ft. by 10 ft. deep, in which it is hoisted (Fig. 14) vertically to the surface and dumped into an ore bin near the top of headframe. (Recently at the No. 1 hoist 364 skips (4,004 tons) of ore were hoisted during a shift of 8 hr.) Shafts 1 and 2

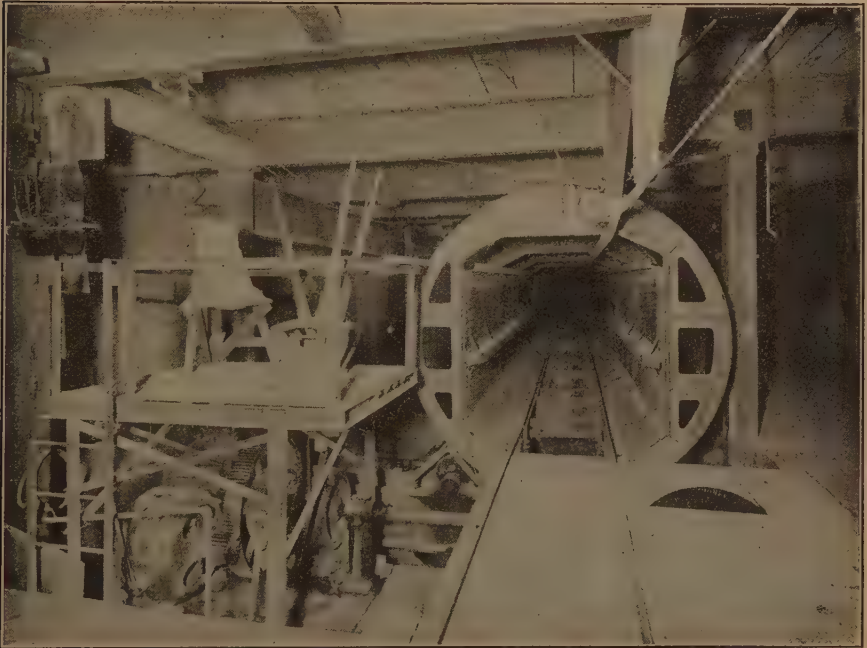


FIG. 12.—TIPPLE, WITH ORE POCKET DIRECTLY BELOW IT.

have two compartments, size 6 by 6 ft. 10 in. The ore then passes through one of four air-lift gates and over a grizzly. The oversize goes to two size 8, style K Gates gyratory crushers, and unites with the undersize from the grizzly, then into a bucket elevator which hoists the ore to the top of building and allows it to pass over an inclined screen chute to two Garfield rolls, size 72 by 20 in. The undersize from the screened chute unites with the crushed ore from the rolls and is fed on to an inclined conveyor belt which takes the ore to the top of a storage bin. (At No. 2 shaft the storage bin holds 5,000 tons while the one at the No. 1 shaft holds 25,000 tons.) From these bins the ore is drawn off through basket gates into railroad cars of 60-ton capacity and shipped to the

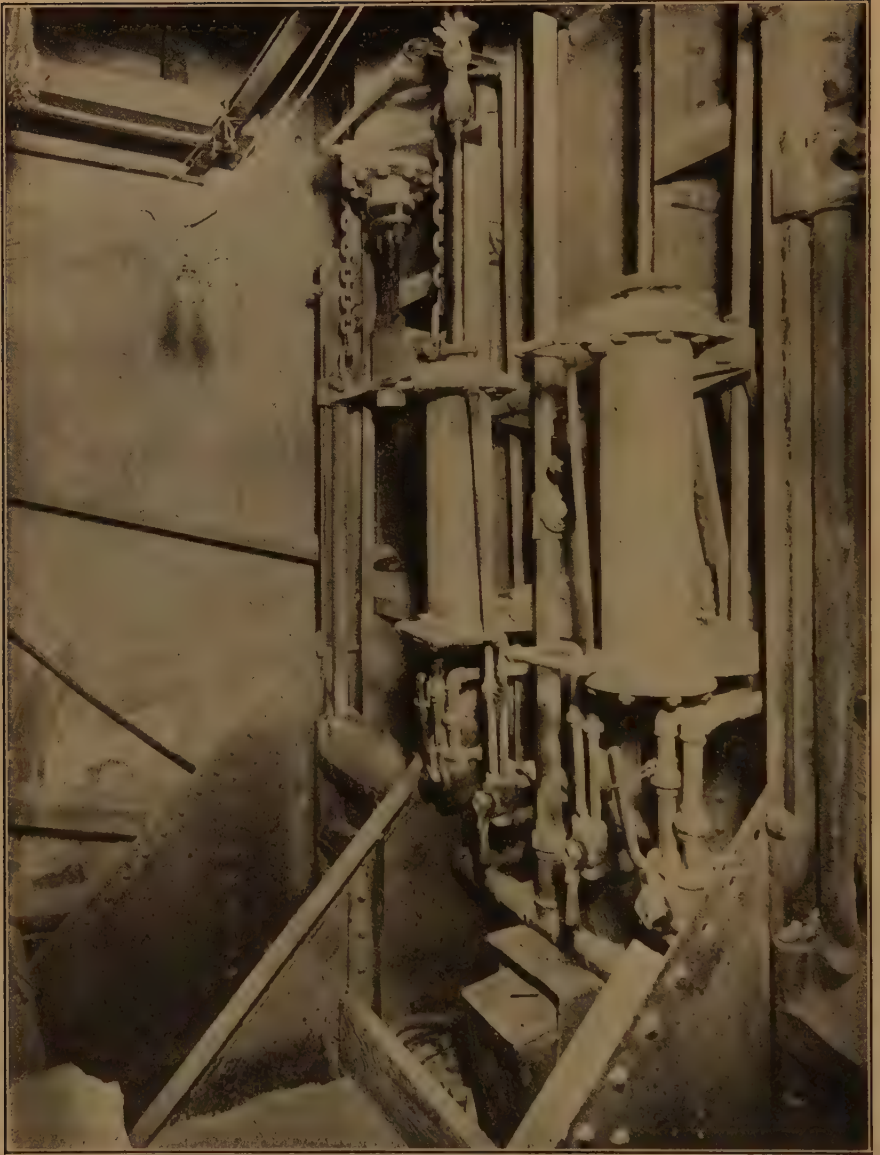


FIG. 13.—AIR-DROPPED GATES, BETWEEN ORE POCKET AND MEASURING POCKET.

concentrator at Hayden, 23 miles from Ray. The ore is usually sent to Hayden in trains of 38 cars, or about 2,400 tons to a trip.

The waste encountered during development and prospecting work is loaded into V-shaped side-dump cars of 45 cu. ft. ($2\frac{1}{4}$ tons) capacity. These cars are hauled by the motors to a winze, and dumped, the waste falling to a pocket directly over an inclined shaft. From the pocket the waste is fed through hand-operated arc gates to skips, size 60 cu. ft. (hoisting capacity $2\frac{1}{2}$ tons). The skips are hoisted to the

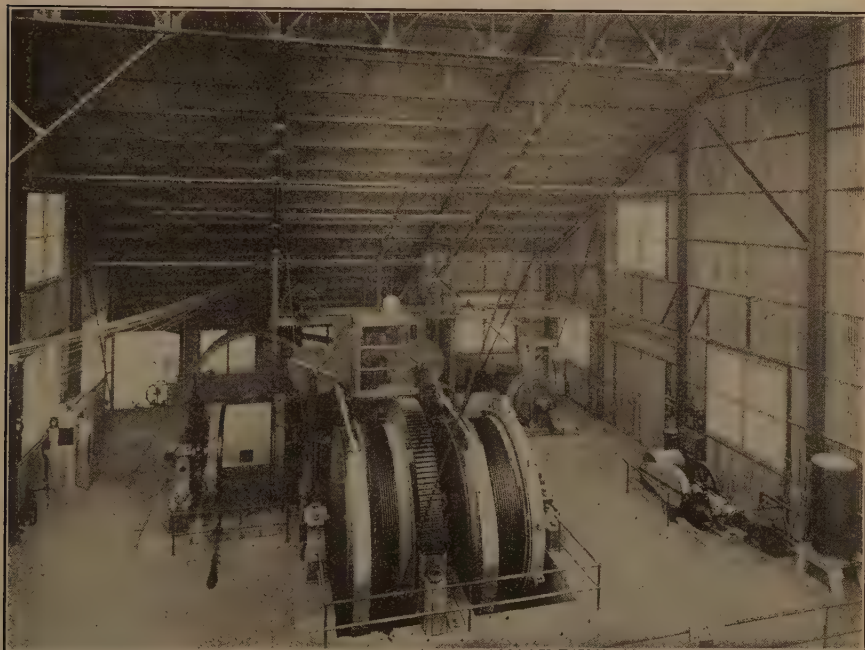


FIG. 14.—THE ELECTRIC HOIST.

top of a 30° incline shaft, where the waste is dumped into a cylindrical iron bin of 10,000 cu. ft. (500 tons) capacity. The No. 1 incline shaft (material hoist) has three compartments, each 5 by 7 ft. Two compartments are used for supplies and waste, while the third contains air pipes, water pipes, electric wires, and a stairway for the men. No men are hoisted or lowered, so that all those working on the lower levels use the stairways (Fig. 15).

The waste is drawn out of the bin into small crab cars and distributed for ballast on railroad, or filling (gobbing) square sets at No. 3 mine.



FIG. 15.—INCLINE OR MATERIAL SHAFT. SHOWING SKIP, TWO HOISTING COMPARTMENTS, AND STAIRWAY; ALSO AIR AND WATER PIPES, AND ELECTRIC WIRES.

ENGINEERING WORK IN CONNECTION WITH THE "RAY SYSTEM"

In efficiently handling the "Ray systems," engineering is of necessity the paramount fundamental. From the time churn drilling is started to the final drawing of a stoped area, the engineering department, in co-operation with the management, must be wary and alert.

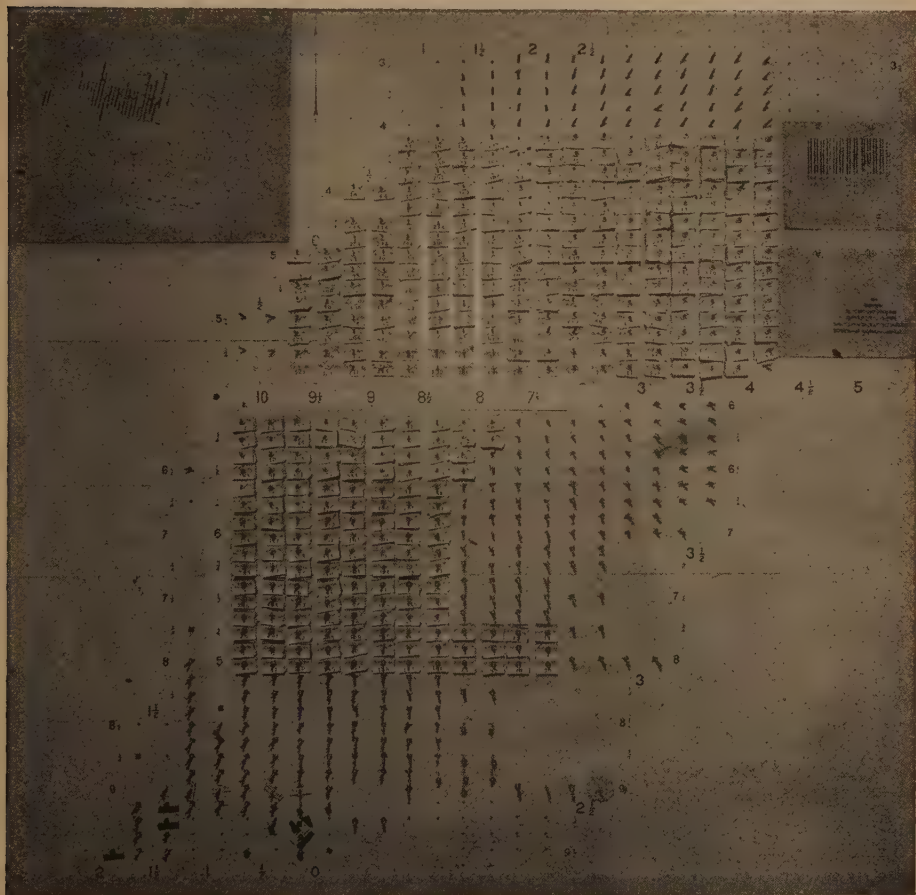


FIG. 16.—CHUTE BOARD. SHOWING THE PADS WITH NUMBERS FOR DETERMINING THE ORE REMAINING IN CHUTES.

Records, maps, and drawings are kept by engineers of each successive step leading up to and during active stoping and reserve drawing of every block of stopes.

A transitman with his helpers is responsible for correct reports on grades, bearings, timbering and proper connecting up of all drifts, inclines, raises, shafts, etc., and plotting of same on maps and sections.

The sampling department takes groove samples at intervals of 5 ft. in all headings, and records assays on maps, sections, and in permanent office record.

"Stope engineering" covers the work of efficiency in underground mining and in making reports and suggestions from time to time for betterment in operation; keeping records of the tonnage obtained and cost of producing ore from each stope, and an aggregate for each block of stopes; making semi-monthly cost and tonnage reports, which are posted on a bulletin board at "Candle House," where all the bosses may see and know the results that each is obtaining in his particular section of the mine; taking samples from time to time along the back of each stope while active so as to keep bosses posted as to whether the stope back is in ore or waste (these assays are entered upon the stope sections); also calculating tonnage and assay value of each block of ore with a proportional distribution of same to each chute to be drawn, therefore by keeping account of all ore drawn from each chute, the amount still remaining is known, and an even settlement of the ore and capping can be effected; keeping the management informed not only as to the surface movement over each area, but also as to the ore remaining in each and every chute in the various sections of the mines by means of "chute boards." These boards are 6 ft. wide, $7\frac{1}{2}$ ft. high and 1 in. thick, with holes 2 in. apart horizontally and $1\frac{1}{4}$ in. apart vertically (see Fig. 16). Each hole represents a chute, and is drilled large enough to receive a bolt $\frac{3}{16}$ in. in diameter. Bolts are $3\frac{1}{2}$ in. long to receive pads of thin paper or tickets which are printed with consecutive numbers. Each ticket is 1 by $1\frac{3}{4}$ in. in size and perforated through the center; the top half contains a hole for the bolt, and the lower half, a number. When ore is drawn from any chute, tickets indicating the amount of ore drawn are torn from the pad representing that chute. A ticket showing the total amount of ore in each chute is placed over the upper half of the pad, so that it is easy to determine the tons of ore still remaining by deducting the number on the lower half from the one on the upper half of the pad representing that particular chute. Figures are placed across the top of a board to designate the drifts, while others down the side indicate the chutes.

Permanent Stope Records are kept by the stope engineers showing the tons produced, together with cost and assay value of same, for each stope, both during active stoping and reserve drawing; also for pillar drifting, cone drifting, and cone stoping. The information for these records is obtained as follows:

A stope card for each class of work (Fig. 17) is turned in to the engineering office by the boss after every shift. Each card shows the labor and material used during that shift chargeable to stoping operations. The items chargeable to stoping operations in the sub-level system are:

1. All work of every description above the first sub-level within the block of ground to be stoped, preparatory for stoping, except that pertaining to manway drifts and stope drifts.
2. Belling and widening out of chute raises.
3. After all active stoping has been completed and reserve drawing begun all work of every description within the stoped area on and above the motor-haulage level.
4. All work pertaining to pillar drifting, cone drifting, and cone stoping.
5. All work of recovering timber after reserve drawing has been finished and section abandoned.

FORM M 21-REV

RAY CONSOLIDATED COPPER CO.

MINE NO. _____ LEVEL _____ STOPE REPORT _____ 191

PLACE OF WORK			SHIFT					
LABOR			MATERIAL				PRODUCTION	
	CONTRACT	COMPANY	POWDER STICKS				CHUTE	CARS
LOADERS			CAPS		FUSE			
MUCKERS			CANDLES		LAMPS			
TRAMMERS			LUMBER	AMOUNT	SIZE	LENGTH		
CHUTE BLASTERS			Pests					
SCREENMEN			Caps					
STOPEMEN			Angle Braces					
PICKMEN			Collar Braces					
			Sills					
			Lagging					
			Standards					
TIMBERMEN	American		Spilling					
	Mexican		Chutes					
			Ladders					
			Kauway Timbers					
TIMBER-HELPERS	American		Other Timbers					
	Mexican							
SAFETY--The co-operation of all employees in our efforts to prevent accidents is earnestly requested.							Signed,	STOPES BOSS

FIG. 17.—STOPE REPORT BLANK.

In the hand-tramming system the charges to stoping operations are:

1. All work of every description above the tramming drifts within the area to be stoped, except manway drifts.
2. After active stoping has been finished and reserve drawing started, all work on the tramming level and sub-level within the area stoped.
3. Cone drifting with the necessary timbering; chute building and cone stoping in same.
4. Recovering all good timber after cones have been drawn and section abandoned.

A car report (Fig. 18) is obtained from the car checkers showing the number of cars drawn from each chute, with a grand total for the section.

Samples are taken of the ore from each chute, by the trammers or car pushers throwing into a box a handful from each car drawn. These samples, after being properly tagged by the car checker, are turned over

Form M-34

CLASS _____

PAGE _____

CAR REPORT
RAY CONSOLIDATED COPPER CO.

MINE No. _____

LEVEL No. _____

SECTION

DATE _____ 191__

SHIFT_____

[illegible]

FIG. 18.—CAR REPORT BLANK.

to the trammers, who are responsible for their proper delivery to the assay office. At noon of the following day an assay sheet is posted in the

engineering office where the car checkers may see the value of the samples taken the previous day and regulate drawing accordingly.

Stope Record Sheets

In the monthly record a separate account is kept for each stope. Each account consists of two pages. One page is used for labor and material, forming the cost sheet (Fig. 19). The other sheet is used for a car record and forms the tonnage sheet. All cost sheets are kept together, likewise the tonnage sheets. Each sheet is ruled to accommodate one month's run. Several other extra sheets are also kept as a summary check on the entries of the others.

All entries on the cost sheets are made in terms of shifts or fractions of a shift under the various labor headings. Material is debited under the proper heading according to the units reported for each class. A price list and wage scale are kept near at hand. At the end of each 15-day period the total is found for each item and multiplied by the price. Cross additions are then made to get total labor and total material. The sums of these give the total labor and material.

The car reports from the various stopes are then entered on the proper tonnage sheet in terms of cars. The total in the right-hand column gives the total cars per shift. To the right of this column the corresponding assay is entered and this multiplied by the total cars per shift gives the cars per cent. per shift, which is entered to the left of the total car column. At the end of each period the vertical columns for each chute are added and their sums must check with the sum of total cars per shift column. Each of these totals is then multiplied by a factor to reduce each from cars to tons. The cars-per cent. column is then footed up and multiplied by the same factor to get tons-per cent. Tons-per cent. is then divided by the total tons to get the proper assay average for the period. Similarly, the total cost of labor and material is divided by total tons on each sheet to get the cost per ton.

As a check on the price multiplication the shift sums from the bottom of each column are entered by stopes on a separate sheet and these correspondingly totaled up. Each sum is then multiplied by its proper rate and the total of this should check the total of the labor and material columns of all the stope sheets. Any error is thus quickly traced.

As a check on the car entries a separate sheet is kept for the total cars per shift as given on each car report. The sum of these totals should check with the sum of the total cars per shift column of all the stope tonnage sheets. The tons-per cent. entries have no independent check, but all assays are carefully entered and all multiplications for cars-per cent. checked by casting out the nines. Average assays and costs per ton are obtained by the slide rule with sufficient accuracy, but are checked

RAY CONSOLIDATED COPPER CO.

418

UNDERGROUND MINING SYSTEMS

STOPE

MINE NO

RAY, ARIZONA

STOPE COST RECORD

LEVEL.....191

Date		Shift	LABOR												MATERIAL										TO DATE	
			Loaders	Trammers	Chute Blasters	Screen Men	Stope Boss	Stopemen	Pickmen	Timbermen	Timber Helpers	Car Checkers	Assaying	Others	Total Labor	Powder	Caps	Fuse	Candles	Machine Drill Exp.	Timber	Total Ma-terial	Total Labor & Material	Production, Tons		
1	1	Day																								
		Night																								
		to Day																								
		Night																								
15	15	Day																								
		Night																								
		Total for Period																								
		Amount																								
16	16	Day																								
		Night																								
		to Day																								
		Night																								
31	31	Day																								
		Night																								
		Total for Period																								
		Amount																								
		Total for Month																								
		Amount																								

Fig. 19.—STOPE RECORD SHEET.

by repeating. So far the checking has only been for the benefit of the office work. As a check on the stope cards themselves, or any missing cards, the various costs for the period are grouped under the headings of:

1. Breaking; 2, tramming; 3, timbering; 4, engineering, etc.

The first includes stope bosses, stopemen, and pickmen.

The second includes loaders, trammers and muckers, chute blasters, and screenmen.

The third includes timbermen and timber-helpers.

The fourth includes car checkers, assaying, engineers, etc.

The sum of these items should equal the total labor as found from all the cost sheets. To this figure is added total material. The final sum must check with the total labor and material from all cost sheets. The totals thus segregated are then comparable with the cost accounts as kept by the clerical force, these being independently made up from the mine timekeepers' reports.

To reduce and concentrate the bulk of these monthly records, a separate cover is used for the summary sheets. The form for these sheets is identical with those used for daily entry. The totals for each column in terms of cost only are entered at the end of each period with the date. A separate sheet is kept for each stope. The upper half of each sheet is used for costs and the lower half for tonnage. On this latter the entries are made in terms of tons and tons-per cent. only. On the extreme right of each sheet are carried the totals to date of labor and material, production in tons, and tons-per cent. The average costs per ton and assay to date are also calculated and entered.

COMPARATIVE ADVANTAGES OF THE TWO SYSTEMS NOW IN USE AT RAY

It must be remembered that the sub-level system is only applicable in sections of a mine in which the orebody covers a considerable area and has a height exceeding 100 ft. The development of the stope laterals is necessarily expensive; the timbering is costly; and considerable expense is incurred in equipping and maintaining the drifts for motor haulage. Unless the orebody is of sufficient area and height to repay the cost of the initial large expenditure, the sub-level system should be abandoned in favor of the hand-tramming system. The great advantage of the motor-haulage system is that it is capable of producing a comparatively large tonnage, not only in the operation of putting up the active stopes in the section, but more especially when reserve-drawing operations are started. The tonnage per shift that can be produced from a reserve in a sub-level section is practically only limited by the efficiency of the electric-haulage system and the capacity of the shaft and pocket equipment. If such conditions are at hand, the sub-level system will produce a large output at a very low cost. However, in sections of the mine where

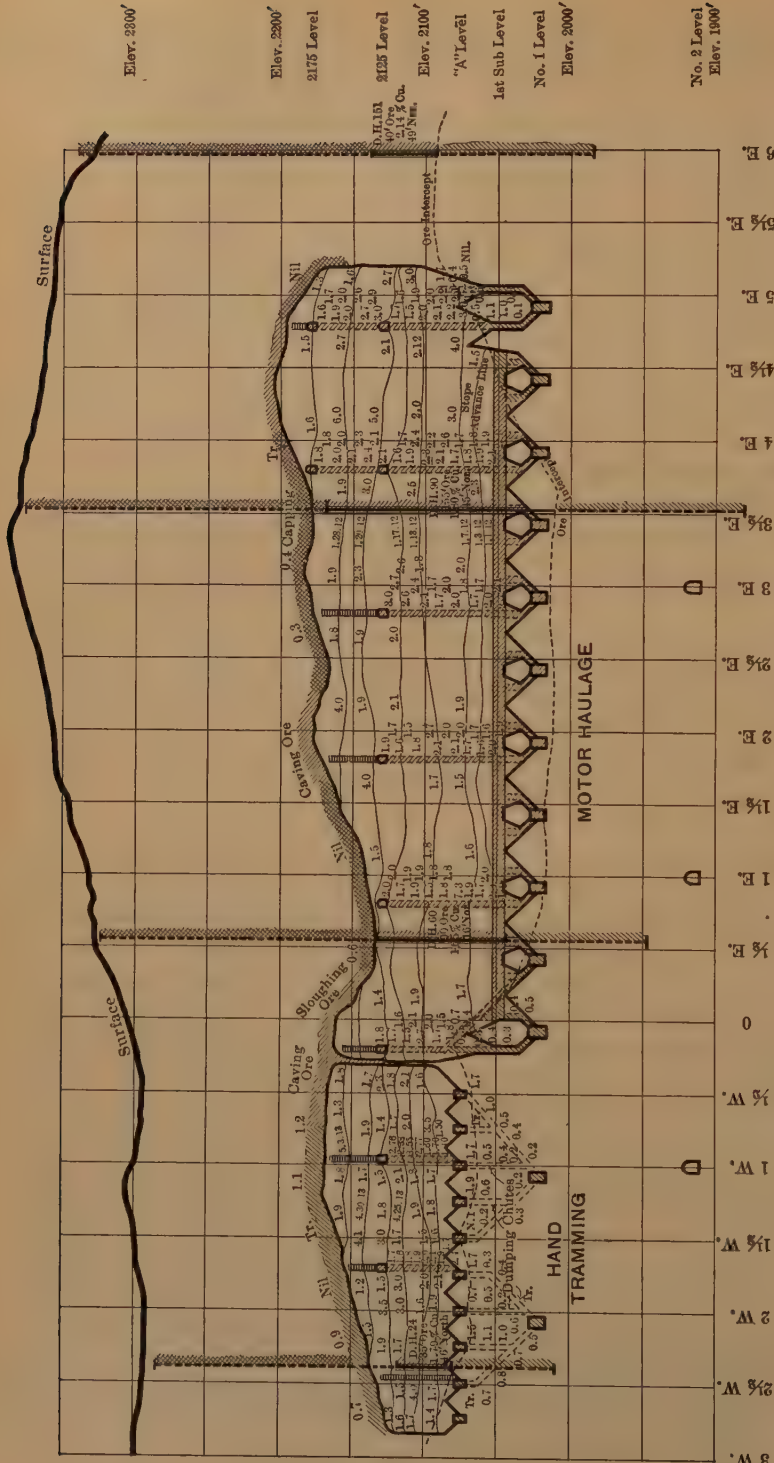


Fig. 20.—SECTION SHOWING PORTIONS OF THE OREBODY ADAPTABLE TO THE MOTOR-HAULAGE SYSTEM AND THE PORTIONS ADAPTABLE TO THE HAND TRAMMING SYSTEM. A SIMILAR SECTION IS KEPT FOR EACH STOPE, SHOWING ORE INTERCEPT, MANWAY-RAISE ASSAYS, STOPE ASSAYS, AND STOPE ADVANCE LINE.

the ore is of less extent and of a height less than 100 ft., the hand-tramming system, primarily owing to the small initial expenditure, will be found more economical to install (Fig. 20). Its advantages over the sub-level system may be enumerated as follows:

1. Less development work is required, with a consequent lower total expenditure; only a small capital outlay is necessary before stoping can be started, and a big tonnage supplied; no great amount of money is tied up for initial expenditures, whereas with the sub-level system all the big expensive motor-haulage drifts must be completed before any ore is obtained from stoping; also the outlay for track, pipe, and trolley wire is high.

2. The total outlay for timber is less, hence the system is adaptable to localities where timber is expensive and difficult to obtain.

3. The total cost for maintaining drifts is lower. When one of the small tramming drifts caves it can be repaired readily with a small outlay, whereas if any of the motor drifts in the sub-level system crush, the repair work is not only expensive, but the time required may result in cutting down the daily output.

4. A higher final extraction can be obtained, as chutes can be built in every set in both the original and cone drifts, thus supplying drawing-off chutes at frequent intervals at the base of the block of ore; but with a sub-level only 30 ft. above the haulage level it is not advisable to cut up the pillar below the sub-level with drawing-off chutes too close together, owing to the danger to the important haulage drifts below; also, the work of running up the raises to the sub-level increases the total cost per ton of ore obtained from a block.

5. The hand-tramming system is adaptable to shallow as well as deep orebodies because of the small outlay for initial development work.

6. The rapidity with which a block can be opened up and made ready for producing ore is an additional advantage of hand tramming.

CONCLUSION

The "Ray systems," although as yet comparatively new in their application, have amply justified all expectations.

The systems have been used in both hard and soft ground. In blocks where the ground is very hard the stopes are carried wider than the regulation 15 ft., thus leaving the remaining pillars narrower and more readily shattered by undercutting. This feature of narrowing and widening stopes according to the nature of the ground insures at all times the safety of the miners, and accounts for the fact that very few accidents occur actually within the stopes. The stopes are under constant supervision, with the backs closely watched and inspected before each drilling.

The method of mining is adapted to the use of stoper (air-hammer)

machines, so that only a few men are needed in producing a big tonnage. All stopes are mined in much the same way and systematically, consequently the men soon become proficient in their work. Since all blasting is done at the end of each shift, but little time is lost. Practically no timber is used above the motor or tramping levels, and in comparison with other caving systems little raising and drifting is done. With the sub-level system even the "stope drifts" on the first sub-level may be done away with by simply connecting over raises direct from chutes on the motor level, as outlined for "Undermining Pillars."

In conclusion, it is but necessary to emphasize the great flexibility of the systems. Practically any number of men can be used during active operations and a corresponding tonnage produced, and when circumstances require it the number of men and tonnage can be either increased or decreased without materially affecting the total direct mining cost per ton.

DISCUSSION

SIDNEY J. JENNINGS, New York, N. Y.—In the historical part of the development of the caving system discussed by Mr. Blackner, I notice the absence of one man's name, Charles Henroten. Mr. Henroten used the caving system in South Africa in the Kimberly mines, and realized from his work there that ground would cave when properly prepared. I think it is due him that his name should be mentioned in connection with an exposition of this system in America.

Mining Conditions on the Witwatersrand

BY W. L. HONNOLD, JOHANNESBURG, SOUTH AFRICA

(San Francisco Meeting, September, 1915)

OWING to a unique labor situation and other unusual circumstances, the mining methods of the Rand are hardly comparable with practice elsewhere. They are of considerable interest, however, and their importance is increased by the magnitude of the industry with which they are identified, gold of \$166,186,000 value, say 40 per cent. of the world's production, having been recovered from 25,701,954 tons of ore milled in 1914.

In the sketchy remarks to follow, attention will first be drawn to some influential factors of general bearing, after which certain features at two mines of the writer's group will be referred to. It is hoped that engineers associated with similar or different conditions and practice will contribute to broaden the discussion.

Before proceeding, recognition is due the generous assistance of C. E. Knecht and F. A. Unger.

GENERAL OBSERVATIONS

The principal mines are situated along the northern limit of an irregularly shaped geological basin, and extend in almost unbroken continuity over a distance of about 60 miles. As the boundaries are defined by vertical planes there are both "outcrop" and "deep-level" properties, all differing in size and in technical features.

So much has been written in regard to the geology of the Rand that only a few pertinent features need here be recalled.

Briefly, the ore deposits are profitably enriched portions of certain conglomerate beds or "reefs," rarely more than three, which are interposed between quartzite, or quartzite and slate, and lie so closely together that where more than one are mined they are dealt with as a working unit, although they may be stoped separately. Their collective extension is locally referred to as "the Reef."

Along the outcrop the series dips more or less steeply, and in depth it flattens, the latter tendency being most marked in the Far East Rand, where the bottom of the basin is nowhere so deep as to be inaccessible.

Dikes and faults are a common feature throughout. They are,

however, so well understood as seldom to involve mistaken procedure, although they may occasion delay and added expense.

The pebbles are mostly of rounded quartz, averaging perhaps less than an inch in diameter, cemented together by a fine-grained quartzitic matrix of metamorphic character. Sedimentary inconstancies, including partings of quartzite, are in evidence, and more or less pyrite is interspersed, which may or may not be appreciably auriferous. Native gold occurs in varying amount, constituting on the average perhaps more than half the assay value of the orebodies.

In the matter of thickness the reefs show distinct local irregularity and differ as to individual tendency and as to the common tendency at particular points. In places they are only a few inches thick, sometimes narrowing to a scarcely perceptible seam with or without scattered pebbles, while in other places they exceed a thickness of 10 ft., a rough average being, say, 33 in., which may or may not include quartzite partings. Thickness, however, is of secondary importance and may be offset by assay value. Hence the common adoption of the "inch-pennyweight" as a unit of measurement, from which the still more significant factor of "value over stope width" may be derived in accordance with the apparent breaking conditions of the locality.

The ore deposits separately considered vary as to occurrence, structural features, and internal and relative enrichment. Along the outcrop the more favored mines have practically no barren territory; in fact, in the Central Rand, where enrichment is unusually persistent, the "leader" has been almost continuously stoped over a length of about 5 miles in the upper workings of several adjoining mines, and on the dip more or less to about 7,000 ft., the deepest development being of a reassuring nature, as will be clear from the subsequent reference to the Village Deep mine. At other mines, and characteristically in certain districts, the bodies, although smaller and erratically disposed, are large enough to be stoped separately, and are so frequently recurrent as to insure fairly stable working conditions. There are also places where considerable areas are rendered profitable through the prevalence of interspersed minor points of enrichment; in other words, the reef is "patchy" but can be worked to advantage as a whole. Somewhat similar irregularity is observable in depth and, although local circumstances may justify considerable confidence, there is always a measure of uncertainty as to the outcome of deeper development at any particular point.

Apart from the irregularity last referred to, it is noticeable in the deeper workings of today that the ore is largely confined to a single reef and that where the conventional series is present the main-reef leader is the more persistently enriched. A corresponding tendency is also revealed in the Far East Rand, where the gold is associated with a somewhat different geological sequence, several reefs being more or less profitably

enriched near the outcrop but only one to a considerable extent in the deeper mines.

As to enrichment in depth various views have been expressed, influenced perhaps by dissimilar experience, and reflecting probably different standards of comparison. It seems sufficient in the circumstances merely to state that, in the writer's opinion, the average quantity of gold per unit of reef area lessens in depth and the ore deposits show more instances of lower than of higher grade. Decreasing average grade, however, is not the only factor involved, nor is it of the relative importance usually assumed, as will be inferred from the following references to deep development in the two more prominent districts.

The Village Deep mine, located in the Central Rand, has at a horizontal distance of about 7,000 ft. from the outcrop reached a vertical depth of 4,600 ft., the dip in the lower levels averaging nearly 40°, with a steepening tendency. Although the tenor of enrichment is inferior to that of mines nearer the outcrop and the South Reef is of decreased importance, the development on the leader in the lower levels still discloses practically continuous ore, of good stoping width and of better than the average grade of the mine, for over 5,000 ft. along the strike, the full extent in this direction, as well as in depth, being still undetermined.

In the Far East Rand, Brakpan Mines has reached a vertical depth of 4,100 ft. at a distance of nearly 4 miles from the outcrop. The formation, therefore, has flattened, local dip averaging about 7°, and the tonnage per claim being accordingly reduced. Only the contact reef, lying at the base of the series on a slate foot wall, is of material importance, and of its total tonnage about 50 per cent. has so far been of profitable grade. The orebodies are clearly segregated, irregular as to occurrence and shape, generally elongated in the direction of dip, of good thickness, and large enough to be stoped separately, some measuring over 700 ft. along the strike and more than 4,000 ft. on dip. Although they vary in the matter of enrichment, both internally and comparatively, and show a general tenor below that of neighboring mines nearer the outcrop, yet their average is approximately that of the Rand of today.

An even more interesting case is that of Springs Mines in the same district, since its workings are 3½ miles from those of its nearest neighbor and about 7 miles from the outcrop, the vertical depth being 3,750 ft. The mine is still in the development stage, but has been laid out on such broad lines as to make the showing of considerable significance. So far, the reef has been opened over an area of about 60 acres, the maximum dip and strike dimensions being respectively 6,200 and 2,500 ft. Characteristics similar to those at Brakpan are disclosed, except that the dip is flatter and that the grade of the ore is exceptionally high, the ore reserve now averaging 10.3 dwt. over an average assumed stoping width of 52 in.

The two last-named mines are specially significant because of their

bearing on the district offering the greatest scope for new activity. It is, however, because of their bearing on the question of ore deposition at great distance from the outcrop, and their influence on mining methods, that they are now referred to. And it is gratifying to find that, although they may afford less profit per unit of area than some of their neighbors nearer the outcrop, they disclose sound mining conditions and promise to justify the heavy capital expenditure involved. Obviously it is imperative in such circumstances that the surface area be ample and not overvalued, that the shafts be so disposed and the layout so conceived as to insure rapid opening in all directions, and that the available capital be sufficient to carry the venture through an extreme phase of poor development or other initial misfortune.

Although the three mines mentioned may not accurately reflect all the existing evidence as to enrichment in depth, they serve to elucidate the opinion previously expressed and to suggest that the future of the Rand is perhaps more dependent on the quantity than the quality of the ore yet to be developed.

The reefs generally, as well as the associated quartzite in somewhat less degree, are above the average of vein quartz in hardness, and the formation is notable also for its cohesion and tenacity. In depth these characteristics are partly neutralized by increasing pressure. It is probable, however, that any consequent advantage to stoping is more than offset by the accentuated weakening influence of dikes and faults and the decreased labor efficiency due to higher temperature and greater humidity. There is, as a rule, but little underground water, and, on the whole, working conditions are comparatively favorable.

Some outstanding correlative factors will now be referred to.

Imported supplies are expensive owing to the high rates of transportation, particularly railage. Imported timber costs about \$1 per cubic foot, drill steel about \$150 per ton, and other imported items are proportionately costly. Explosives, locally manufactured of imported material, are on the basis of \$7.33 per case for gelignite containing 60 per cent. of nitroglycerin and \$10.33 for 92 per cent. blasting gelatin. For inferior mine timber of local growth about 32 c. per cubic foot is paid. Water for industrial purposes is pumped to some extent from the mines, but mainly from surface dams at small cost, and in exceptional cases is purchased from the Rand Water Board at 80 c. per 1,000 gal., which is also the rate for the domestic supply.

In the matter of power, the Rand, apart from the cost of plant, is fortunate. Local coal of about 12.5 lb. bomb test is delivered to individual mines at an average of about \$2.80 per ton, and to centralized power stations at an even cheaper rate. The latter are important factors in the situation, as is shown by the fact that the principal company

is now distributing about 460,000,000 units of electric power over 224 miles of primary transmission line, and 120,000,000 units of compressed air through 30 miles of primary piping. Its charges are on the basis of electric power and average about 1c. per kilowatt-hour. Consequently, the cost of hoisting, pumping, machine drilling, and ventilation is fairly low.

The importance of ventilation and the allaying of mine dust by means of spraying devices has become increasingly apparent in recent years, and the quality of the underground air is now much better than formerly. It is hoped that the measures adopted, coupled with the wider use of hollow-steel drills and greater precaution on the part of employees, will materially improve the position as to phthisis, the economic significance of which is indicated by the fact that about \$3,000,000 per annum is now paid in compensation and about \$50,000 for the maintenance of a sanatorium.

Only a few words seem called for on the subject of labor. Broadly speaking, white employees do but little actual work, their function, which they jealously guard, being to supervise the tasks of the natives. The latter are of various African tribes, differing more or less as to minor characteristics. Generally, they are tractable and fairly dependable, but deficient in dexterity and initiative. They are averse to continuous employment and come to the mines for comparatively short periods, the average stay being about seven months. Hence their progress as workmen is not sustained and standards of efficiency are low. Some, of course, are of exceptional character and capacity and, as a whole, they may in time show greater proficiency than is so far evident. It has to be borne in mind, however, that white labor is unsympathetic to native advancement and that owing to this restrictive influence radical change in labor conditions is improbable, or at least so speculative a matter as to be beyond the scope of the present paper. The relationship between the two classes of labor will be sufficiently apparent from the context. Underground wages vary for different classes of work but average about \$5 per day for whites and 50 c. per day for natives. To the latter rate has to be added about 25 c. per day to cover recruiting, transportation, housing, food, and medical attention.

BRACKPAN MINES, LTD.

The reef characteristics and significant bearing of this mine have already been mentioned. When first considered its remoteness from known conditions rendered it a speculative venture and, even when \$188,000 had been expended on boreholes and other preliminaries, the outcome was problematical. Thereafter a further expenditure of \$5,772,000 distributed over $8\frac{1}{2}$ years, with consequent deferment of

interest, was called for before the commencement of milling. Perhaps no more notable instance of the confidence which the Rand has inspired is available and, in the light of tendencies elsewhere disclosed, it is doubtful if there has been an issue of more vital importance to the industry.

The dimensions of the property are about 6,000 by 14,000 ft. and the shafts are located as symmetrically as considerations of depth and ventilation would allow. Reference to Fig. 1 will show that the south (No. 2) shaft is centrally situated and that it is to the dip of the north or upcast shaft. This disposition is favorable to rapid development, ventilation, pumping, and underground work generally.

Each shaft has seven compartments and measures 42 by 9 ft. overall, the large dimensions being due to ventilation rather than to hoisting considerations. In the north shaft the reef was reached at a vertical depth of 3,098 ft., and in the south shaft at 3,707 ft., both being continued to provide for bin and loading facilities, as well as tail-rope sheaves. Full particulars of the shafts and of the methods adopted in sinking, also cost details, were given in a lecture delivered by Charles B. Brodigan at the Royal School of Mines, London, on Jan. 27, 1913, and published by the *Mining and Scientific Press*, under the title *Rand Practice in Deep Shaft Sinking* (Apr. 26 and May 3, 1913).

Both shafts are provided with steel headgears 105 ft. high, the combined bin capacity of the latter being about 3,700 tons. The ore is transported by means of a locomotive in 40-ton railway trucks from the north to the south shaft, a distance of 4,300 ft., where it is discharged on to belts which deliver it to the sorting and crushing plant. The connecting railway, which is of standard gage, extends to the shops, stores, etc., and also to the main line.

Hoisting equipment is installed for only two compartments of the north shaft. The drums, measuring 11 ft. diameter by 8 ft. face, are directly driven by a 1,450-hp. motor with Ward-Leonard control, 2,100 hp. being absorbed in starting. Main and tail ropes are of $1\frac{1}{2}$ in. diameter and the skips carry a net load of 5 tons.

The south shaft has four compartments equipped for hoisting. Two are served by a Ward-Leonard set identical with that at the north shaft and are used solely for the handling of men, tools, and supplies. The cages have three decks and accommodate 60 men each, about 2 hr. being required for lowering or hoisting the day shift of about 110 white and 1,650 native employees. The other two compartments are practically restricted to the handling of ore. Load, ropes, and drums are unaltered, the drums in this case, however, being directly driven by a three-phase 2,100-volt motor of 1,450 hp., using the standard type of liquid controller. Post brakes operated by deadweight and relieved by air pressure are provided in each instance, together with devices for the automatic prevention of overwinding and tooth clutches for adjustment.

About 150 tons per hour are handled over a vertical distance of 3,850 ft., and the cost per ton, as in the case of the Ward-Leonard at the north shaft, averages 17 c. when hoisting a total of about 64,000 tons per month.

The arrangement of the station at the bottom of the south shaft will be clear from Fig. 2. It will be noticed that after the trucks are knocked off the various haulages they pass in one direction over the bins, where they are emptied by means of hand-operated revolving tipples. The grade of the station roadways is 1 in 60 and of the curves 1 in 50, the trucks traveling with but slight assistance. The bins, which discharge into Kimberley loading chutes, do not call for description. Nor is it necessary to refer to the arrangements at the bottom of the north shaft, except to mention that an 84-in. Sirocco fan is installed, together with doors and other requirements of ventilation.

Development

Fig. 1 shows the extent to which the mine had been opened at the end of 1914, and makes it clear that the main incline was a matter of first importance. The average dip of the section between the shafts is 7° , its length is 4,500 ft., and the width about 18 ft. This section was driven from both ends and was connected at a point 3,173 ft. from the north shaft, the horizontal error being 6.5 in. and the vertical 0.7 in. Almost throughout, the reef was above the incline, but never more than 40 ft. Stations were cut at intervals of 500 ft. down to the 7th level; below that point, owing to the flatter dip, they were spaced about 800 ft. apart.

It will be recognized that in the development of flat-reef mines such as Brakpan certain features assume a more serious aspect than where the dip is steeper. For instance, the accentuated influences of minor undulations and dislocations of reef render it difficult to avoid excessive sinuosity in driving, which not only adds considerably to development costs by reducing the effectiveness of footage but also interferes with efficient tramming later on. And, obviously, comparatively small faults exert an exaggerated bearing on the length of crosscuts. To meet obstacles of this nature straight-line work is often resorted to, although even this is not free from difficulty if the reef is so disturbed as to require frequent changes in the grade, for the removal of broken ground then becomes more laborious, winches having to be placed at the high points, and the handling of water is complicated by the need of pumps at the low points. The latter handicaps, of course, may also be encountered in pilot drives and haulage-ways, but as to these there is no alternative, since their expeditious advancement, as previously pointed out, is a matter of vital importance in such a mine. It should perhaps be added that the bad

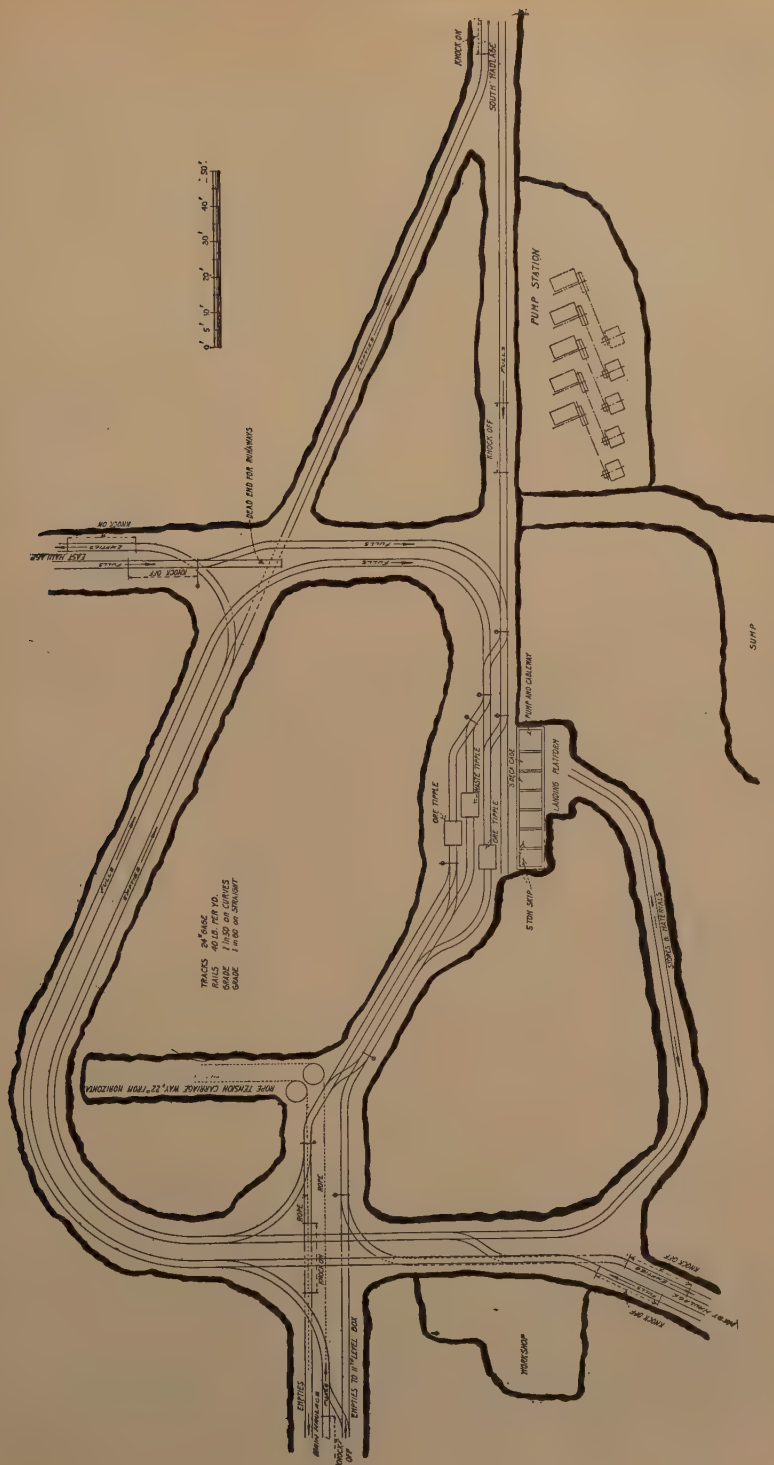


FIG. 2.—ARRANGEMENT OF STATION AND TRACKS, NO. 2 SHAFT OF BRAKPAN MINES.

features referred to largely disappear when the roadways are finally completed and equipped.

The details of development work do not differ materially from common practice. Headings are driven on reef where conditions permit, faults and undulations, however, calling for many departures from the rule. Levels are about 6 by 7 ft.; winzes are carried 9 ft. wide in order to expedite the opening of stopes; main haulageways are 14 ft. wide, and those of secondary importance 11 ft.

Large machine drills are used, mostly of the reciprocating type, with $3\frac{1}{4}$ -in. cylinders. These, however, are gradually being replaced by large hammer-type machines using hollow steel through which water is fed to the bottom of the hole. A high-power explosive is called for and it is found that blasting gelatin, containing 92 per cent. nitroglycerin, gives the best result. Up to the end of 1914 the practice in single-width headings was to blast the cut first, then to turn on a water spray close to the face for about 30 min. to eliminate dust and fumes, and finally to blast the round, all in the same shift and as soon as possible without undue risk to health. In the wider headings, cut and round were blasted simultaneously. Recently the government regulations were so amended as to make it illegal to blast the cut and round separately in the same shift except under special conditions and precautions against dust and fumes approved of by an inspector of mines. Consequently, as far as possible, two single-width headings are now assigned to each miner and the cut and round are blasted on alternate shifts. The full effect of this procedure on costs and the rate of progress is not yet clear.

Drives are equipped with single track of 20-lb. rails laid to 24-in. gage, and haulageways with double track of 30-lb. rails as well as the necessary endless-rope fittings. Both are electrically lighted and provided with air and water pipes of ample size.

The cost of development during 1914, exclusive of shoveling, tramming, winding, and pumping, but inclusive of equipment, amounted to \$15.87 per foot for drives, crosseuts, raises, and winzes, and \$25.60 per foot for haulageways.

Stoping

Whether hand or machine stoping, or a combination of both, should be adopted in flat-lying mines is a question on which opinions may differ. It is not entirely a matter of natural conditions, since certain associated factors have to be taken into account. In favor of machine stoping it may be urged: (1) that the generally existent shortage of native labor calls for the adoption of mechanical means as far as possible; (2) that hand-stoping natives show less efficiency in such a mine as Brakpan because of their aversion to the "uppers" or dry holes frequently required near the roof; (3) that in most stopes the reef is of fairly good

thickness and its value therefore does not suffer excessively through the increased stope width inevitable with machine mining; (4) that the cost of machine mining is lower, particularly in the wider stopes referred to; and (5) that it will probably enable the supply of a larger average tonnage to the mill. Advocates of hand stoping may claim: (1) that the system requires smaller capital expenditure; (2) that a somewhat narrower stoping width is obtained; and (3) that there is less shock to the roof and consequently less demand for support. Where both methods are in use it is, of course, possible to vary their proportionate employment according to the ebb and flow of native labor and thereby to render the output somewhat less fluctuating.

At Brakpan the unusual course of all-machine mining was decided upon and an equipment of 375 drills capacity is now installed. That the decision was warranted is shown by the fact that the mine has seldom had its full quota of native labor, notwithstanding that the total number of machines on the Rand has since increased by more than 50 per cent. Operations are more sensitive to labor fluctuations than in mines that are partly hand stoped, but this is a matter of secondary importance and, in some measure, is regulated by increasing or decreasing the accumulation of broken ore in the stopes.

Reciprocating machines of $2\frac{3}{4}$ -in. and $3\frac{1}{4}$ -in. cylinder diameter are now mainly used in stoping, the smaller size being restricted to stopes under 60 in. and accounting for about 22 per cent. of the tonnage broken. Small drills of the hammer type with automatic feed and hollow steel are, however, being tried with marked success and promise in time to replace the reciprocating machines throughout. Two natives are attached to each machine—a hammer boy and a spanner boy—and there is a native drill carrier for every two machines. They work under the supervision of a white miner who is on contract and who, as a rule, has charge of four machines irrespective of size. Large machines show an average efficiency of 0.62 square fathom, equivalent to 11 tons, broken per machine shift, and small ones 0.49 square fathom, or 7 tons, the average stope widths being 70 in. and 57 in. respectively. The poor showing, as compared with that obtained elsewhere, is due to labor conventions, white and native, which restrict the daily task per machine, whether large or small, to approximately four holes of 6 ft. depth. The higher efficiency of the large machines may therefore be taken as a measure of the more favorable breaking conditions in the wider stopes.

About 57 per cent. of the explosive used in stoping is blasting gelatin, containing 92 per cent. nitroglycerin, the remainder being gelignite, containing 60 per cent. nitroglycerin. Generally the stronger explosive is used in the narrower stopes and the weaker in the wider ones; individual stopes, however, may call for a reversal of the rule.

The procedure in breaking ore is influenced by the dust factor and the

consequent importance, as well as the advantages, of drilling wet holes. It is therefore the practice when using machines of the reciprocating type to resort to underhand stoping and, when using hammer drills with hollow steel, to back stoping. In opening a stope, work is usually started at both the bottom and top of the winze, which, as already noted, is 9 ft. in width. A winch is placed at the top to deal with the upper portion of the stope, and another half way down to raise and lower the trucks that serve the bottom section by way of the level below. The benches at first conform to diagonals from the winze to the levels, giving the appearance of underhand stoping at the top of the winze and overhand at the bottom, but after reaching a distance of about 100 ft. on either side their lateral extension is checked and they are advanced mainly in the direction of the winze until the stope is opened throughout. Each face is then more or less parallel to the winch track, from which laterals have been thrown off at intervals of 35 ft. for extension as required. "Backs" vary from 300 to 1,200 ft., according to the influence of dip on the distance between levels. Stope pillars of good size are left where necessary, especially along fault lines, and these, together with frequently placed "pig-sty" packs, sufficiently support the roof. As the faces advance right and left the roof of the interior gradually subsides, the weight being taken by the packs. This may or may not call for the shifting of winch tracks. Serious falls of roof seldom occur and the isolation of the orebodies precludes regional movement and therefore renders sand filling or other permanent support unnecessary. After a stope has been mined out and cleaned, unless there are special reasons for keeping it open, the pillars are drawn, a large proportion of the pig-sty poles are recovered, and the roof is allowed to completely collapse. Rock packs are seldom resorted to in the stopes but, in combination with rock walls, are generally employed to protect levels and haulageways that may have to be maintained through worked-out stopes.

Fig. 3 shows one face of a stope that has been well opened, including the track arrangement to be subsequently referred to.

The cost of stoping during 1914 was, for large machines 98c. per ton broken, and for small machines \$1.20. After allowance for the included ore from development faces and other sources, and for the waste rock discarded in sorting, the general average on a per-ton-milled basis was \$1.03.

Shoveling, Tramming, and Underground Haulage

This subject may be divided into two stages. The first covers necessary handling at the working face, hand tramming to the winch track, hoisting or lowering in trains to a level, and hand tramming to a haulageway. The second covers delivery by means of endless-rope haulage to a shaft bin.

Fig. 3, which has previously been referred to, shows the general arrangement of tracks and the position of winches in a stope that has been so far advanced as to call for the shifting of winch tracks. Explanation seems unnecessary.

Rigid trucks, of low design and adapted to the tipples, are used. They hold $20\frac{1}{2}$ cu. ft., equivalent to one ton of ore, have a height of 2 ft. $11\frac{3}{4}$ in. from top of rail, and a length over buffers of 5 ft. 3 in. The truck body is of steel and the bottom is curved to a 17-in. radius. Bottom and ends are of $\frac{3}{16}$ -in. plates and sides of $\frac{1}{8}$ in., a stiffening ring 2 by $\frac{1}{2}$ in.



FIG. 3.—IDEAL PLAN OF STOPE, SHOWING GENERAL ARRANGEMENT OF TRACKS, ETC., BRAKPAN MINES.

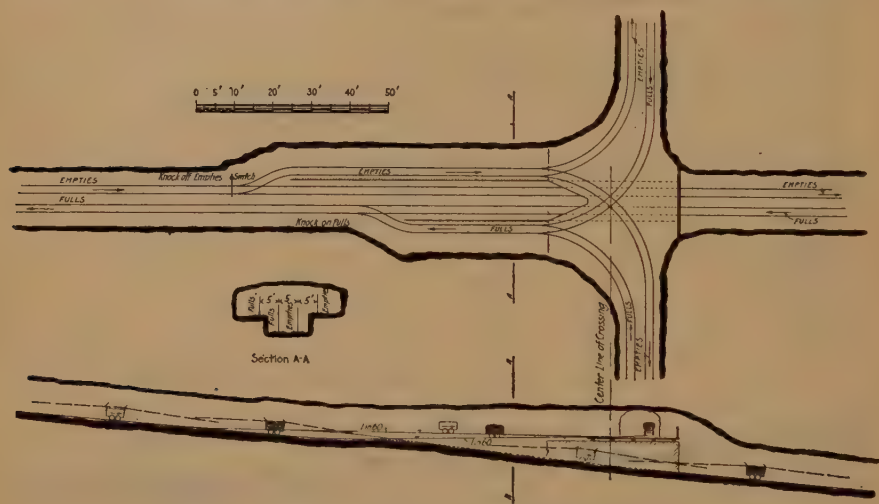
running around the top and greatly adding to the general rigidity of the body. Brackets, suitable for the Cradock type of overhead clip, are riveted to the ends, which are stiffened by a $\frac{3}{16}$ -in. plate on the inside. The underframe is built of 6 by 4 in. oak, cut to fit the cylindrical bottom of the body, to which it is securely bolted. Cast-steel wheels of 15-in. diameter are used, one being fixed and the other running loose on a 2-in. steel axle which is fitted with a cast-steel grease sleeve. The wheel base is 18 in. and the gage, as previously stated, 24 in.

Of the natives working on shoveling and tramming, 43 per cent. are loading boys employed at the face. They are on piecework and receive from 8 c. to 10 c. per truck, which includes loading and delivery to the

main winch track, and they handle somewhat over 6 tons per shift of 8 hr. The remainder includes boss boys, winch drivers, rope boys, track boys and level trammers, the last named constituting about one-fourth of the total. These are on day's pay, averaging about 50c. per shift.

The cost of shoveling and tramping in 1914 was 68 c. per ton trammed. This figure includes all expenditure from the working face to the haulage and covers development as well as stoping. It is unfavorably influenced by the fact that this class of work offers the best opening for inexperienced and surplus natives.

In the initial stage of production all ore was hand trammed to the main incline, but with increasing tramping distances the supplementary system of subsidiary haulages has had to be gradually brought into



[FIG. 4.—15TH LEVEL STATION OF BRAKPAN MINES.]

operation and is still in course of extension. Fig. 1 shows the present position in this respect. It will be noted that the main incline and the main incline south follow the longitudinal axis of the property and conform more or less to the dip of the reef. Together with the east incline and the west incline they feed into the station at the bottom of the south shaft. See Fig. 2.

Fig. 4 is a plan of the 15th-level station, where an eastern haulage joins the main incline south, and Fig. 5 shows the arrangement, as far as completed, at a point between the shafts from which traffic may be directed either upward or downward. Stations for hand-trammed traffic are similarly arranged in accordance with circumstances. Four natives are usually required for knocking trucks on and off, the rope being lifted by hand out of or into the clips.

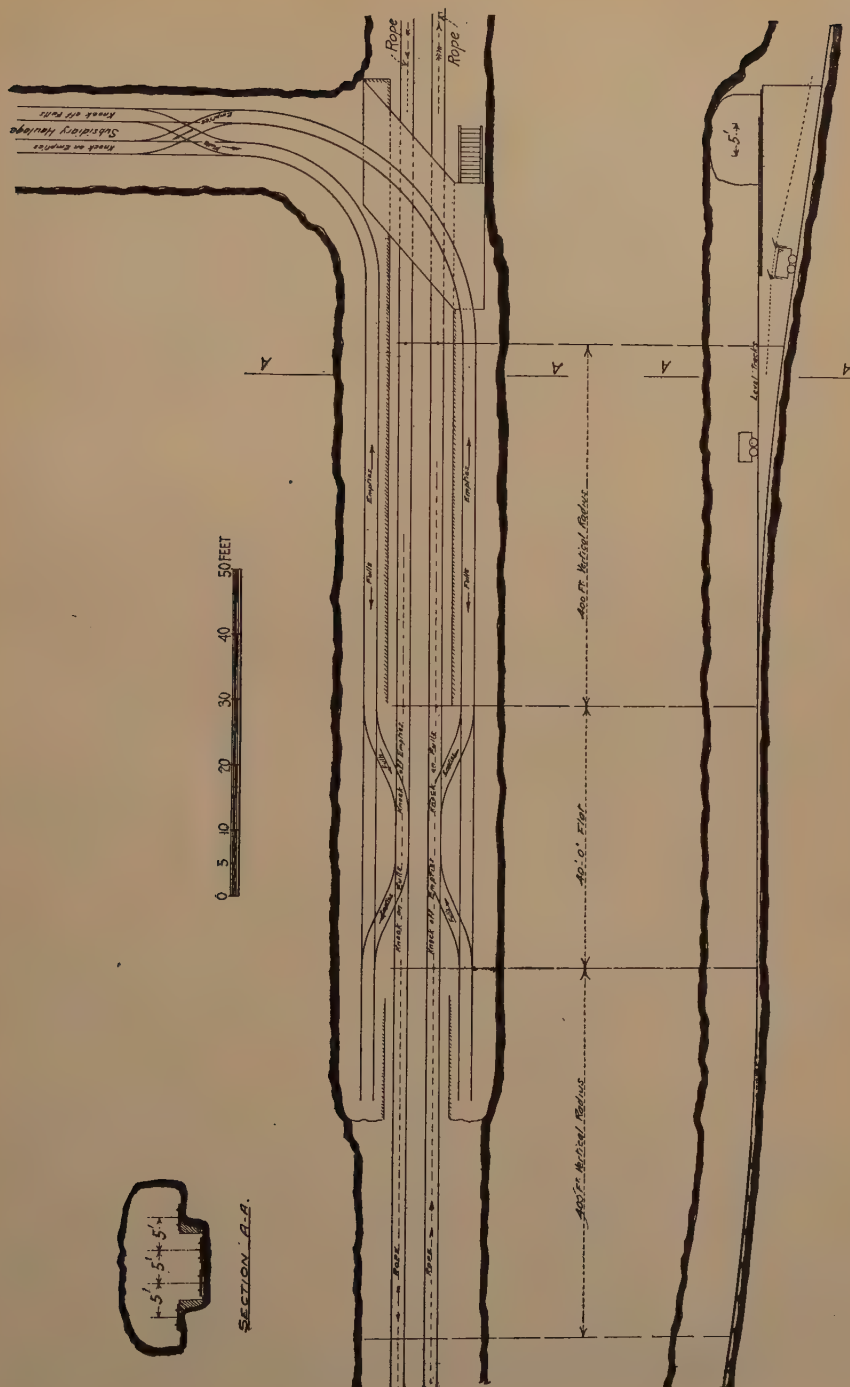


FIG. 5.—ARRANGEMENT SHOWING JUNCTION OF SUBSIDIARY HAULAGE WITH MAIN INCLINE, BRAKPAN MINES.

All the haulages, as previously mentioned, are provided with double tracks of 30-lb. rails, laid on timber sleepers to 24-in. gage, and the necessary endless-rope equipment. The gears are electrically driven and designed for a rope speed of 2 miles per hour, at which rate little trouble is experienced from runaway trucks or other mishaps.

The cost of operating all haulages in 1914, during which period a total of 705,775 tons was delivered to the shaft bins, averaged 10 c. per ton, about 75 per cent. of the amount representing white and native labor, 5 per cent. power, and the remainder ropes, fittings, workshops, and sundry stores. This also covered the transportation of tools and supplies. Obviously, in comparatively considering the problem of underground haulage, it is necessary to take into account the mileage involved, as well as hoisting and surface transport.

Ore Reserve and Grade Control

Stopes, as well as development, are regularly sampled in detail, and at the beginning of the year the ore reserve is computed. Previously included "blocks" are then recalculated, if their faces have been altered by stoping or further development, and the new tonnage is taken in at its estimated value.

Owing to the elongated form of the orebodies a block between adjacent levels can be estimated with fair accuracy before winzing. A third face justifies greater confidence, but, if provided by a winze or stope in the central and richer portion of the body, it may somewhat exaggerate the average assay value and render it necessary to apply an "assay-plan factor" in calculating the probable recovery value of the ore reserve. No such correction, however, is called for in respect of total gold, since overestimates of the assay value are offset by underestimates of tonnage, due to the fact that development does not always expose the full width of the reef. The lateral extent of the bodies is seldom sufficient to warrant more than one winze; hence blocks with four faces exposed are rare.

The correct estimation of stope width is a matter of great importance because of its bearing on tonnage and recovery value. Each block has to be carefully considered and an allowance made for the inches of hanging wall and foot wall which will probably be included in stoping. Experience has shown that the combination of a flat reef and machine mining will seldom permit of stopes less than 48 in. in height and consequently this figure is taken as a minimum, even though the reef may be abnormally thin. Experience has also shown that, regardless of the thickness of reef, at least 15 in. will be broken outside the "reef channel" and that a further allowance may have to be made because of "slips" to which the

ground will inevitably break. Taking the mine as a whole, the average stope width exceeds the average reef width by about 22 in.

The details of calculation need not be referred to, as they involve customary steps and precautions. In the end the results are compiled so as to show the numbered blocks in consecutive order, their area on dip, reef width and value, stope width and value, and tonnage. The blocks of profitable grade are then classified and cast under three headings, depending on whether they are available for stoping, in process of becoming available, or locked up in shaft or other pillars. A record is also kept of the blocks that cannot be worked to advantage in existing circumstances. It is found in practice that, on further opening, the latter frequently provide a considerable tonnage of profitable ore.

About 73 per cent. of the ore broken comes from the ore reserve, 17 per cent. from development faces, and 10 per cent. from other sources. The last-mentioned sources include selected portions of old blocks previously classified as unprofitable, new blocks not yet taken into account, pillars, hanging-wall leaders subsequently discovered, and odds and ends from old workings. Ore falling under this category is usually of lower grade than that from the ore reserve. This is still more marked in respect of development, since it is not easy to eliminate the rock from the poor faces, and even the product of the good faces suffers because of the abnormal proportion of waste which has to be included in breaking. The factors just referred to, coupled with sorting practice and milling extraction, enter in some degree into all forecasts of future recovery. A typical calculation may perhaps simplify the matter. Assume:

	Per Cent.		Dwt. per Ton
	73	From ore reserve	6.7
	17	From development	2.8
	10	From other sources	5.6
	100	to sorting plant	5.98
Deduct	14.5	excluded by sorting	
Deliver	85.5	to mill	6.93
Recovery, with 96 per cent. extraction, say,			6.65

Clearly, any variations in the assumptions would correspondingly affect the result. It is necessary, therefore, in making a forecast, either to put forward a definite figure as a measure of existent conditions in the light of probability, or to give limiting figures sufficiently comprehensive to cover the various contingencies which may arise. Among the factors of doubtful bearing may be mentioned the uncertainty as to native labor, since variation in the supply may alter the relative quantity and quality of ore from the three sources mentioned, and possibly the percentage of

sorting. The position and outlook as to development may also enter. Furthermore, there is always a chance that the grade may vary more than anticipated owing to an unusual combination of the irregularities of enrichment. Of course, the number and the widely differing values of the faces afford an equalizing influence, as well as a means of adjustment, but not to the extent of insuring persistent uniformity.

In order to guard against undue depletion of the richer areas, as well as of those more favorable to cheap working, a monthly analysis of the blocks mined is prepared. This includes the fathomage and proportion taken from each block during the month, its assay value and width as determined by current sampling, and the assay value and width of the blocks as at the time of the last calculation of the ore reserve; also totals and averages. Comparison of the average assay value and width of ground stoped with corresponding figures of the total ore reserve discloses whether the ore broken was of mine grade and width. And, of course, any over- or under-mining is apparent.

Comparisons on a yearly basis are also interesting in that the monthly fluctuations are eliminated and the accuracy of ore-reserve estimates is more clearly defined. They show that as a rule the average of stope assays is slightly below that of the blocks, the winzes, as previously explained, probably exerting an undue influence on block sampling. But, as against this, they also show that the average reef width determined by stope sampling exceeds the corresponding figure for the blocks. The latter gain, as already stated, is probably due to the incomplete exposure of the reef at certain points along the levels, and the net result in practice is that the total gold recovered per unit of area closely agrees with the amount indicated by the ore-reserve estimate.

SPRINGS MINES

The remarks previously made as to the characteristics and significance of Brakpan Mines are, generally speaking, applicable to Springs Mines and, together with the specific references already made, sufficiently indicate the outstanding features of this property.

Briefly, the position is that, after six years' work and an outlay of over \$4,000,000, the mine is now ready for equipment and other expenditure that will involve \$1,500,000 to \$2,000,000 additional and fully two years' further deferment of interest. Fortunately, as already intimated, the showing of the development so far accomplished is remarkably good.

Fig. 6 will show the outlines of the company's holding, which totals 1960, acres, as well as the disposition of the shafts and the development accomplished at the end of 1914.

The shafts are similar to those at Brakpan. They have been de-

scribed, together with some particulars of conditions and procedure, in a paper by B. D. Bushell entitled Notes on Sinking Operations at Springs

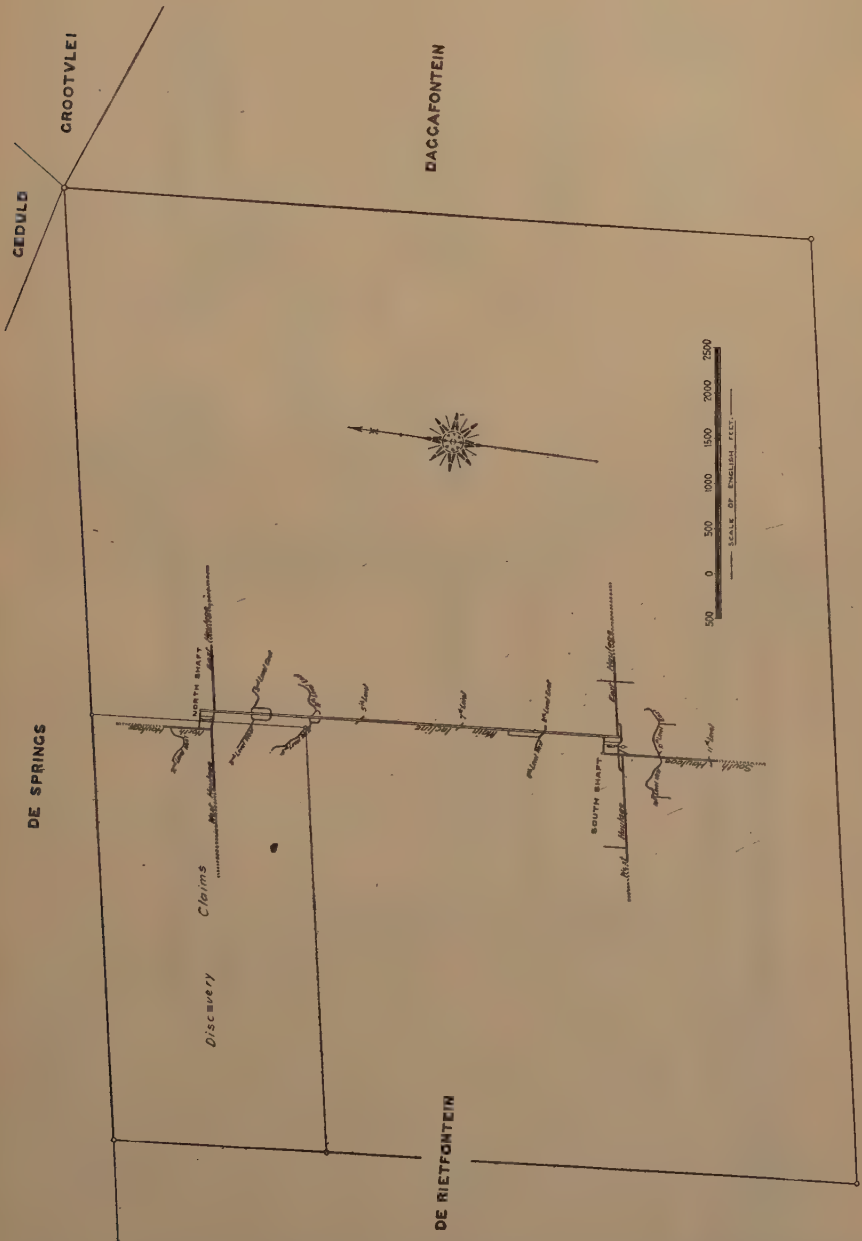


FIG. 6.—PLAN OF UNDERGROUND WORKINGS OF SPRINGS MINES.

Mines, which was discussed by the Institute of Mining and Metallurgy, London, at a meeting held on May 22, 1913. The outstanding feature of

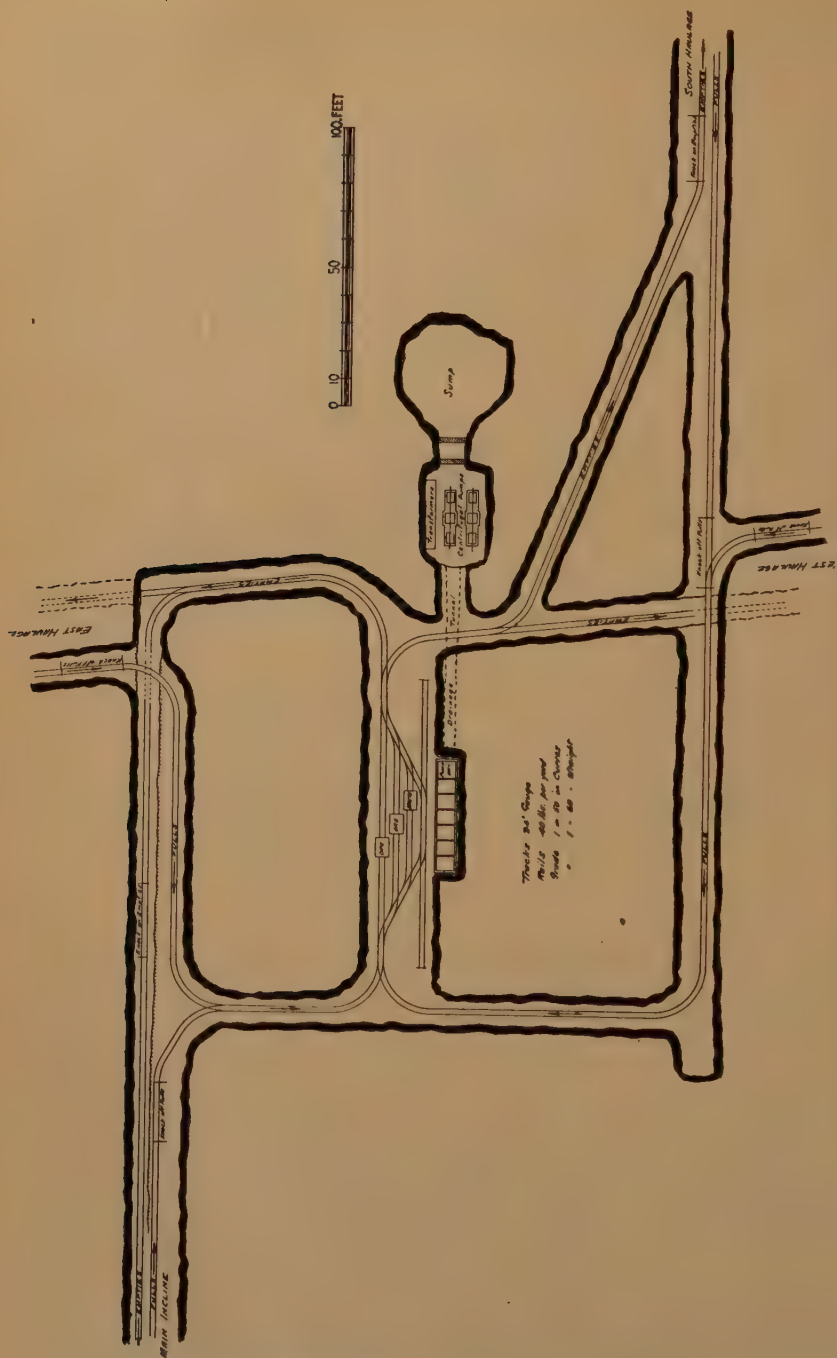


FIG. 7.—ARRANGEMENT OF STATION AND TRACKS, SOUTH SHAFT, SPRINGS MINES.

interest was the quantity of water dealt with in passing through the dolomite. In the north shaft it was first encountered at 460 ft., and it could not be "caught up" until after the Witwatersrand quartzite was reached at 605 ft. For a period of seven months the average rate of sinking was only 27.3 ft. per month and the average quantity of water hoisted was over 2,000,000 gal. per day.

Haulageways are necessarily an important feature of the underground layout. It will be noted that six are already well advanced, and now that the main incline has been connected it is the intention to start two others from its middle point as soon as possible. In view of the Brakpan

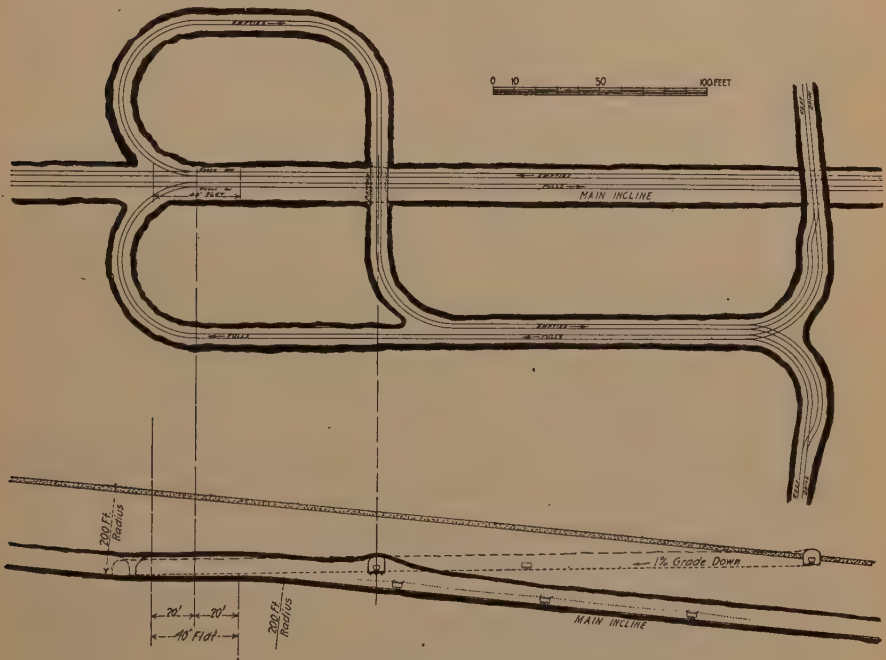


FIG. 8.—8TH LEVEL STATION OF SPRINGS MINES.

experience it was felt that the flatter dip of the reef called for still greater recourse to straight-line development. Fortunately but little water has been encountered, notwithstanding the unusual quantity in the dolomite above and the fact that faults and dikes often occur. Progress, therefore, has as yet been unimpeded and, although water may still be met with, the haulageways are now so far advanced that serious inconvenience is improbable. It is because of the absence of water and the need of economy that the sump shown in Fig. 7 is still so small.

Further reference to Springs seems uncalled for at present, Figs. 6, 7, and 8 sufficiently indicating the principal departures from Brakpan procedure.

Churn-Drilling Costs, Sacramento Hill

BY ARTHUR NOTMAN,* M. E., BISBEE, ARIZ.

(San Francisco Meeting, September, 1915)

SACRAMENTO HILL is a mass of granite porphyry intruded along a fault between Paleozoic sediments and pre-Cambrian schists in the Bisbee district, Cochise County, Arizona.

The intrusion invaded both walls of the fault to a large extent, and the accompanying extensive mineralization of the Paleozoic limestones is the locus of the great high-grade copper orebodies of the Bisbee mines. Prior to 1909, although the presence of ore in the mass of the hill itself had long been suspected, systematic exploration had not been attempted. A few shallow pits on the surface here and there had disclosed some chalcocite with associated carbonates as stringers in a mass of strongly brecciated, silicified, and iron-stained porphyry. The workings in the surrounding limestones had penetrated the porphyry in a dozen or so places for some distance from the contact but had disclosed no ore. However, the rock had been largely altered to quartz and sericite and very thoroughly impregnated with pyrite and traces of the copper sulphides. A close geological study of the conditions, both surface and underground, strengthened these suspicions, and plans for prospecting the hill were made in 1909.

Reference to the accompanying illustrations, Figs. 1 and 2, will give an idea of the general appearance of the surface. The mineralization is more intense in two main zones, one at the west end and the other at the east end of the hill.

The outcrop of the hill is elliptical and approximately 3,000 ft. long by 1,400 ft. wide.

The Sacramento shaft, which is the main hoisting shaft of the mine and situated at the east end of the hill, as shown by the accompanying map (Fig. 3), offered a convenient base for prospecting that end of the hill, while the workings from the Holbrook shaft offered a similar opportunity at the west end. The choice of level for the first work was determined by analogy with the most favorable depth for enrichment below surface observed in the other disseminated deposits of the Southwest.

* Chief Geologist, Copper Queen Consolidated Mining Co.

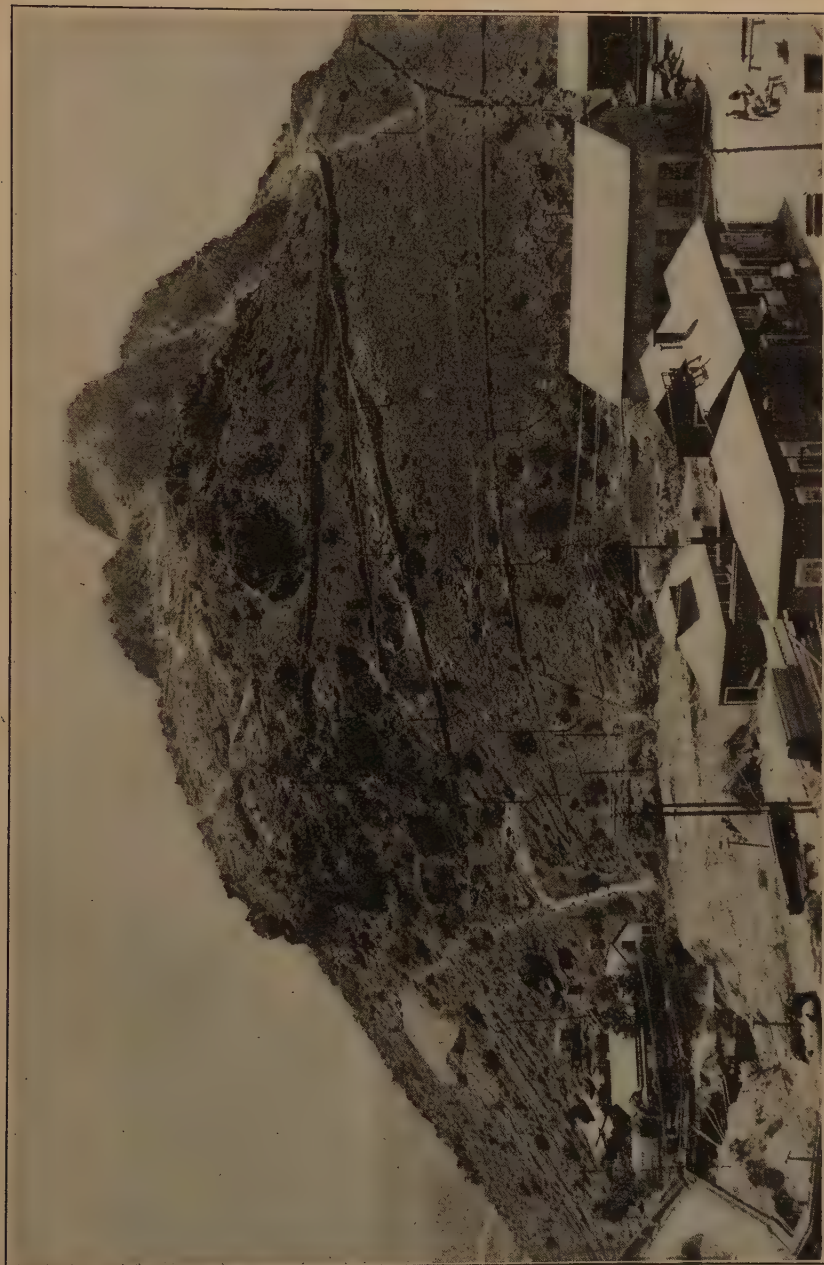


FIG. 1.—WEST END OF SACRAMENTO HILL FROM A POINT NEAR THE HOLBROOK SHAFT.



FIG. 2.—SOUTHEAST SIDE OF SACRAMENTO HILL, SHOWING THE SACRAMENTO SHAFT IN THE CENTER, THE TEST MILL AT THE RIGHT, AND THE POWER HOUSE AT THE LEFT.



FIG. 3.—MAP SHOWING LOCATION OF CHURN-DRILL HOLES AND UNDERGROUND DEVELOPMENT ON SACRAMENTO HILL.

Accordingly, drifts on the 400-ft. level of the Sacramento shaft were started north and south (436 drift—see map) to crosscut the most favorable surface showings. This work disclosed about 90 ft. of mineralized porphyry, containing disseminated pyrite, chalcopyrite, and chalcocite, which averaged 2.0 per cent. copper by careful channel sampling. At the same time, 277 drift from the Holbrook workings was started east under the long axis of the hill to serve as a base from which to crosscut the best surface showings at the west end of the hill. This drift failed to disclose any commercial mineralization in the porphyry. Following this work, a tunnel was driven in from the north side of the hill, over 436 drift, which disclosed oxidized and leached porphyry with the exception of about 40 ft. of enriched material, averaging slightly under 2 per cent. A station was then cut in the Sacramento shaft, halfway between the tunnel level and the 400-ft. level, and a drift, 3-158, run north, which resulted in the exposure of 210 ft. of material averaging 2.21 per cent. copper. A crosscut from 277 drift, 2-131 drift (see map), was started and encountered 25 ft. of material averaging 3.8 per cent. copper.

The work of development was then undertaken in earnest, and up to February, 1913, when underground work was discontinued, a considerable tonnage, averaging 3.53 per cent. copper, was estimated to have been developed by 19,604 ft. of drifts, crosscuts, and raises.

At this time it was decided to continue the development by means of churn drilling. Both the lower cost of development and the greater desirability of the vertical sections of the ore obtained by the drills, as compared with the horizontal sections obtained with underground methods, were factors influencing this decision. The company had been operating a Star drill on somewhat similar ground. The costs obtained exclusive of the cost of the drill, were as follows, for 8,000 ft. of drilling:

	Cost per Foot
Labor and supplies	\$1.99
Roads	0.71
Total	\$2.70

The general plan adopted for the drilling of Sacramento Hill was outlined as follows:

1. The drilling of a line of holes to crosscut the area of the hill between the two well-mineralized areas at either end. The purpose of this work was to determine at the earliest moment the approximate extent of the orebodies.

2. *a.* In case the orebodies were proved continuous by (1), the whole area of the hill was to be drilled at the corners of 200-ft. squares, with

intermediate holes where the irregularity in outline of the ore seemed to require it.

b. In case this work proved the central portion of the hill to be barren, the work was to be laid out on 100-ft. squares, with the ore developed underground as starting points. Here, too, when it seemed necessary, intermediate holes were to be driven.

With these objects in view, a road was constructed to drill the line of holes called for by (1). These holes showed no ore, and consequently the plan described under (2-b) was adopted and the roads laid out on the ground to cover the orebodies and their probable extensions with 100-ft. squares, with the minimum amount of road building.

The cost of the road building for the period considered was as follows:

Labor and supplies.....	\$22,065.77
Footage of road.....	15,092
Cost per foot.....	\$1.02

About this time, owing to the expense of delivery of the coal, and the proximity to the central power house (see map), it was decided that some cheaper power could be secured. The condensation losses eliminated the question of piping steam from the power house. Compressed air was barred out by the fact that the pressure of 100 lb. to the square inch at the power house seemed too low for the purpose. Delivery of fuel oil to the drill by pumping from the storage tanks at the power house was favorably considered, but finally electrical power was decided upon as offering the most advantages and the fewest disadvantages. Accordingly, two No. 22 Armstrong drilling rigs without engines or boilers were secured (see Fig. 4) and equipped with motors of the following specifications: 25 hp., 230 volts, 94 amperes, 400 to 1,200 r. p. m.

At the same time, an arrangement for steering the machines when moving was devised, and installed on each machine. The moving was accomplished by means of "dead men" and the drilling rope and reel. This arrangement was somewhat less convenient than that of the steam drill, which was self-propelling. A transmission line 2,860 ft. long, of the following specifications, was installed from the power house at the following cost:

2,860 ft. cable; 0000 rubber-covered standard.

1,500 ft. cable, 00 flat mining machine (cut into 250-ft. lengths).

Labor.....	\$486.69
Supplies.....	1,434.74
Total.....	\$1,921.43



FIG. 4.—SIDE VIEW OF ONE OF THE ELECTRIC DRILLS IN OPERATION.

From the main line, which was run approximately parallel to the axis of the hill, temporary lines of the flat cable were run to the drill locations.

Tables I and II give the costs obtained in this work. The figures for No. 1, or the steam drill, cover a period of 22 months, during which time 12,153 ft. were drilled in outside ground, and 16,617.5 ft. in Sacramento Hill. The figures for the Nos. 2 and 3 electric drills cover a period of only 3½ months.

TABLE I.—No. 1 Churn Drill (Operated by Steam)

	Cost	Cost per Foot	Per Cent. of Cost.
Labor:			
Drilling.....	\$13,497.62	\$0.469	33.5
Casing.....	2,113.50	0.073	5.2
Fishing.....	564.86	0.020	1.4
Moving.....	5,679.03	0.197	14.1
Repairing.....	3,488.94	0.121
	\$25,343.95	\$0.881	62.8
Supplies:			
Coal.....	3,819.22	0.133	8.7
Casing.....	1,336.16	0.046	3.3
Drilling tools.....	2,880.33	0.100	7.1
Cordage.....	2,334.01	0.081	5.8
Miscellaneous.....	1,779.90	0.062	4.4
	12,149.62	0.422	30.1
Storehouse.....	126.11
Water line.....	413.57
Air line.....	16.95
Steam line.....	275.20
	831.83	0.029	2.1
Teaming.....	2,000.00	0.000	5.0
Total.....	\$40,325.40	\$1.402	100.0
Footage drilled, 28,770½			
Footage drilled per 8-hr. shift,			
1.17			
Initial costs.....	3,734.87		
	\$44,060.27		

TABLE II.—Nos. 2 and 3 Churn Drills (Operated by Electricity)

	Cost	Cost per Foot	Per Cent. of Cost
Labor:			
Drilling.....	\$3,867.45	\$0.466	32.7
Casing.....	1,215.25	0.146	10.3
Fishing.....	597.24	0.072	5.0
Moving.....	1,696.82	0.204	14.3
Repairing.....	698.06	0.084	5.9
	\$8,074.82	\$0.973	68.3
Supplies:			
Casing.....	170.39	0.021	1.4
Drilling tools.....	548.56	0.066	4.6
Cordage.....	770.40	0.093	6.5
Miscellaneous.....	1,610.81	0.194	13.6
	3,100.16	0.373	26.2
Electric power.....	276.22	0.033	2.3
Teaming.....	370.65	0.045	3.1
Total.....	\$11,821.85	\$1.425	100.0
Footage drilled, 8,294			
Footage drilled per 8-hr. shift, 15.2			
Initial costs.....	6,578.75		
Electric power line:			
Labor.....	486.69		
Supplies.....	1,434.77	1,921.46	
Grand total drills Nos. 2 and 3..		\$20,322.06	
Grand total drill No. 1.....		44,060.27	
Assaying cost.....		1,132.59	
Roads:			
Labor.....	21,055.55		
Supplies.....	1,010.22	22,065.77	
Grand total.....		\$88,412.41	\$2.385
Total footage, 37,064½			
Average depth of hole, feet, 427			
Footage of roads, 21,814			
Cost per foot of roads.....	\$1.01		

The following table gives the cost for the steam drill on Sacramento Hill alone:

TABLE III.—No. 1 Churn Drill (Sacramento Hill)

	Cost		Cost per Foot	Per cent. of Cost
Labor:				
Drilling.....	\$7,696.85		\$0.463	41.5
Casing.....	1,261.84		0.076	6.8
Fishing.....	253.51		0.015	1.3
Moving.....	2,939.66		0.178	16.0
Repairing.....	1,732.93		0.104	9.3
		\$13,904.79	\$0.836	74.9
Supplies:				
Coal.....		1,702.63	0.102	9.1
Miscellaneous.....		1,867.51	0.112	10.1
Water lines.....		237.39	0.014	1.3
Teaming.....		850.00	0.051	4.6
Total.....		\$18,562.32	\$1.115	100.0
Footage drilled, 16,617½				
Footage drilled per 8-hr. shift, 20.4				
Teaming estimated.				

It is not quite fair to compare the figures in Table III with those for the electric drills, as the larger part of the work by No. 1 was done in nearly barren ground of uniform character, while that of the electric drills was done entirely in ore-bearing ground, where the changes from strongly silicified leached capping to soft enriched ore, and harder primary material, rendered the drilling much more difficult. For a three months' period, in which the drills were working under essentially similar conditions, the following figures were obtained:

Operating Expense

	No. 1 Steam	No. 2 Electric	No. 3 Electric
Cost per foot.....	\$1.34	\$1.56	\$1.15
Footage drilled.....	4,169	3,198	4,510

Analysis of the working conditions shows that No. 2 drill was operating in exceptionally bad ground and that, while not conclusive, these figures indicate that eventually the electric drills should be operated at from 10c to 25c. per foot cheaper than the steam drill, under the conditions obtaining.

The work was conducted as follows: It was carried out under the immediate supervision of a drill superintendent, and operations were continuous in three 8-hr. shifts, excepting on Sundays. Each shift on each machine consisted of one driller, one helper, and one sampler. There was also a chief sampler, who was responsible for the sampling on

all drills. On day shift, there was one team with a driver and helper for approximately half a shift to deliver coal for the steam drill, wood for drying samples, to haul supplies, repair parts to and from the machine shop, casing, samples, and miscellaneous equipment when the drill was moving from one hole to another. The electric drills were equipped with portable forges operated by electric motors, while the steam drill was equipped with a forge operated by a small steam turbine. Both smithing coal and coke were used in these forges.

The routine duties of the various members of the crew were as follows: The driller was in charge of and operated the machine; the helper oiled the machine, dressed bits, and assisted and relieved the driller when necessary; the sampler arranged for the sampling of the sludge every 5 ft., the drying of the sample, the tagging and recording of the sample, and its transportation to the assay office. When not engaged in sampling, he was at the call of the driller for repairing, bit setting, or moving. The chief sampler, one-half of whose time was devoted to the supervision of underground sampling, had general charge of the drill sampling, and kept the office records up to date.

In the costs given, the sampler's time is included in the labor charge, as only a small portion of his time is really chargeable to sampling.

The tools used were standard and are of no particular interest in a discussion of costs, excepting that lack of proper tools, or a sufficient number of them, is undoubtedly a most frequent cause of high labor costs in churn drilling. The labor and supply charges for casing depend entirely on the character of the ground drilled and so comparisons of these items for the two kinds of machines are of no value. The time lost in fishing depends on the character of the ground, the supply of tools, and the nature of the power. Experienced drillers operating a steam drill are able to tell immediately when there is danger of the tools sticking by the character of the exhaust, and by taking necessary precautions can avoid undue strain on the drilling rope, and possible loss of the tools. In the case of the motor-driven drill, this becomes a matter of much more delicate observation, and to inexperienced men offers considerable difficulty. This is indicated by the increased loss of time in fishing with the electric over the steam drills, and the slightly increased cordage expense.

The labor of moving is practically the same with both types of drill. The repair labor item for the steam drill is higher, due partly to inherent disadvantages and partly to the fact that it was a second-hand machine.

In considering the cost of power on the two types we encounter the widest difference in the cost of operation. Unfortunately, the cost of teaming on the steam drill, at least 60 per cent. of which is chargeable to the coal, is not very accurate, owing to the fact that stable expense in the past has been very roughly segregated. This would indicate a cost of



FIG. 5.—THREE ARMSTRONG ELECTRICALLY OPERATED CHURN DRILLS ON THE WEST END OREBODY. THIS VIEW SHOWS THE CHARACTER OF THE ROAD BUILDING.

17½ c. per foot for the coal and another 2½ c. for pipe lines, or a total of 20 c. per foot.

Electric power cost only 3.3 c. There is no teaming to be charged against it, while the proper charge for the installation of the power line depends on the amount of drilling to be done. For a minimum of 30,000 ft. this would amount to 6.4c. per foot, or a total of 9.7 c. per foot, or less than one-half the power cost of the steam drill.

The 4 c. difference in the item of drilling tools in favor of the electric drill is more apparent than real, inasmuch as the steam machine was put into commission more than a year prior to the starting of the electric drills and some of the tools then purchased have been used on all machines.

The 13 c. difference in miscellaneous supplies in favor of the steam drill, on the other hand, is due to the fact that a considerable portion of this item might very reasonably have been charged to initial costs.

The initial costs include the purchase price of the machines, the freight, and, with the exception just noted, all additional permanent equipment.

The road building completed is sufficient for a total of 60,000 ft. of drilling to an average depth of 400 ft., or a cost of 36.8 c. per foot of hole drilled.

The assaying cost amounts to 3.5 c. per foot.

The roads built are 9 ft. wide, excepting at turns, where they are 12 ft. wide. The maximum grade, of which there are two short stretches, is 30 per cent. There was a great deal of rock work and dry walling to be done of a character shown in Fig. 5. In general, the roads follow the contours of the hill.

The initial costs of the three machines, distributed over a minimum of 60,000 ft., amount to 17.2 c. per foot.

These figures would indicate the following complete costs for 60,000 ft. of drilling:

	Cost per Foot
Initial costs.....	\$0.17
Operating expense.....	1.43
Road building.....	0.37
	<hr/>
	\$1.97

A salvage of 7 c. a foot on the initial costs might be reasonably allowed and it seems probable that the operating expense can be lowered another 5 c. to 10 c. per foot, giving an average cost of about \$1.80 per foot for a minimum of 60,000 ft. of drilling.

The checking of the churn-drill sampling, which after all is the most important part of the operation, has not progressed very far as yet. One raise has been driven on a churn-drill hole in ore with the following results:

Assay of Samples

	Churn Drill, Per Cent. Copper	Channel, Per Cent. Copper
	3.66	4.00
	3.72	2.50
	4.18	3.40
	2.38	5.60
	5.78	4.50
	4.25	3.70
	2.82	4.70
	3.30	2.80
	2.90
Total.....	34.21	34.10
Average.....	3.80	3.79

The channel samples were taken in 5-ft. lengths, about $2\frac{1}{2}$ ft. from the drill hole. The weight of sample taken was about 8 lb. per foot. The elevation of the top of the ore, as shown by the drill-hole record, was checked exactly by the raise; the hole, however, continued in ore below the level from which the raise started, so that the elevation of the bottom of the ore has not yet been checked. The results indicated are highly satisfactory.

The office records of the samples consist of longitudinal sections and cross-sections through the drill holes, showing the assay value of each 5-ft. sample, on which the probable outlines of the orebody are drawn, using all information available from both underground development and churn drilling. From these sections, the tonnage developed is calculated.

The conclusions warranted by the costs obtained so far, are

1. That 60 to 75 per cent. of the operating cost is labor.
2. That the labor item is materially affected by the supply of tools, and the accessibility to a machine shop for heavy repairs.
3. That the choice of power in churn drilling cannot affect the costs obtained, excepting to a very minor extent. In the present case, the low cost of electrical power, due to the proximity to the central power house, determined the selection.

Tramming and Hoisting at Copper Queen Mine

BY GERALD F. G. SHERMAN,* BISBEE, ARIZ.

(San Francisco Meeting, September, 1915)

THE ore deposits of the Warren district, in which the mines of the Copper Queen Consolidated Mining Co. are situated, have been described in a number of technical publications, and will not be discussed here in detail. Certain of their characteristics, however, control methods of underground transportation and hoisting of ore.

In the Copper Queen mine, the majority of the ore has occurred in a zone encircling the west boundary of the porphyry intrusion of Sacramento Hill. It has a width varying from 800 to 1,200 ft., and a thickness of about 400 ft. It reaches the surface in the older part of the mine to the northwest, but dips to the southeast, where it is reached at 1,400 ft. below the Czar collar, in its farthest extension at present developed on Copper Queen ground. There is one major extension from the northwest end of the zone toward the west along the Czar fault, and others of minor importance.

Individual orebodies are scattered through the zone in an eccentric manner, only matched by their own irregularities of form and size. Their most general characteristics are the softness of the ore and their great horizontal rather than vertical extent. It has been estimated that the average vertical thickness of ore in the Czar and Lowell divisions is between 30 and 35 ft. It is calculated by assuming it to be uniformly distributed over its horizontally projected area.

In this zone, and for some distance above it, the ground has been subjected to intense alteration and intense but irregular oxidation. It has resulted in an enormous quantity of earthy or clayey material, which may be either ore or waste, which when wet is both heavy and tenacious. Below alteration, the ground is fairly hard, and the limestones contain primary ores differing from those heretofore considered typical of the camp, and which have not yet been thoroughly exploited.

The mine production has been drawn from many orebodies spread over a great area.¹ A diagram map (Fig. 1), of haulage drifts and loading stations, illustrates the situation in 1914.

*Superintendent, Mines Department, Copper Queen Consolidated Mining Co.

¹ A vertical projection of the ore bodies is shown on p. 509 of this volume in the paper of Charles A. Mitke.

HAULAGE SYSTEM
COPPER QUEEN CONS. MINING CO.

SCALE OF FEET

0 500 1000

LEGEND

Drifts	Chutes
Level	Under 75 Tons
200	0
400	75 to 300 "
600	300 to 1000 "
800	1000 to 1500 "
1000	1500 to 2000 "
1200	2000 to 3000 "
1400	3000 to 6000 "
1600	6000 "

Productive Area.

Sacramento Hill Porphyry

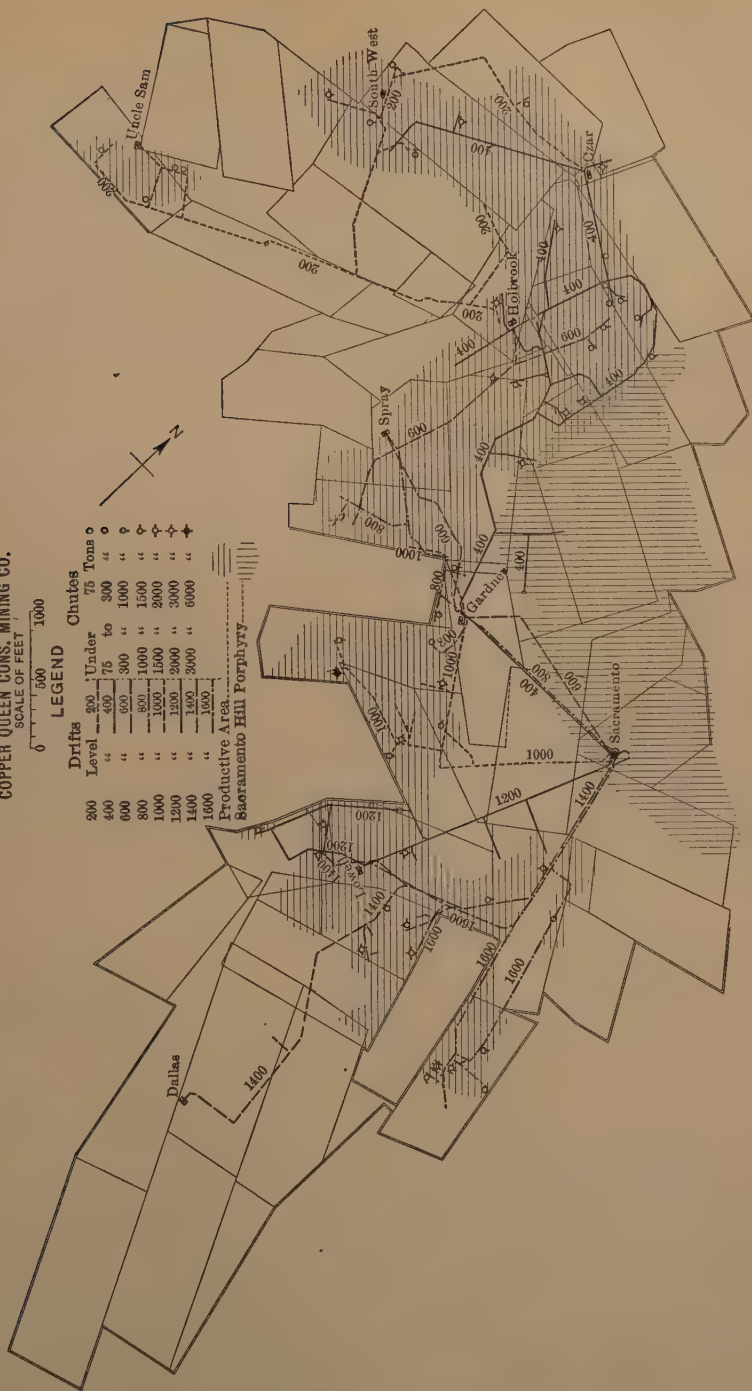


FIG. 1.—HAULAGE DRIFTS AND CHUTES IN 1914.

The effort to restrict the area of work and intensify production has not been particularly successful, for several reasons:

First: The mine has been in continuous operation for more than 30 years. During the earlier period, the costs of mining and smelting were high, and ores which are now profitable could not be worked. Exploration was not carried far enough in advance of mining to explore orebodies as a whole, and they were attacked piece-meal. Much ore was, in consequence, left in and about the old stopes. When new ground was opened to supply the demand for greater production, the old areas were not yet exhausted, their output was merely reduced. In fact, more thorough exploration and the gradually decreasing cost of mining, etc., have actually increased the production and reserves of the Czar division, the oldest part of the mine.

Second: As all ore is smelted direct, a balanced production must be made to maintain a self-fluxing mixture. Since the oldest divisions, the Czar and Holbrook, have contained highly refractory ores, their output has been limited to what the smelter could handle, and their reserves have not been exhausted as rapidly as the economies of mining alone would suggest.

Third: While many of the orebodies are large when taken as a whole, they are small in section, complicated in form and mingled with waste. They cannot be mined rapidly.

The producing area has therefore increased rather than diminished.

All the ore is not sticky or hard to handle, but some is extremely so. To one not acquainted with it, it is hard to appreciate how seriously this quality interferes with every operation of handling, and forces complications into a general plan to insure certainty of action. There is not such a large proportion of sticky ores at present as in the past, except on the 400 level, but a successful method must be prepared to take care of it wherever it appears.

To summarize briefly: The three characteristics of the deposits which affect haulage are scattered operations, heavy ground in and near the ore, and clay ores.

Old Haulage System Unsatisfactory

Prior to the introduction of the system now in use, the ore was trammed by hand to one of several shafts, where it was hoisted on cages and loaded into railroad cars for shipment to the company's smelter at Douglas. In most cases, it was dumped directly into the cars instead of loading from bins, since it was slow and hard work to get it out of storage once it was put in. An attempt was made to mix the ore when loading into the railroad cars, but without great success. Each tenth mine car was loaded into a separate car as a sample for the smelter, but the returns were not beyond question.

An unusual number of shafts were required, since active production covered a wide area. Distances to the shafts were excessive for hand tramming, and it was impossible to maintain such great lengths of drifts and track in proper condition, although heavy repairs were continuous.

The defects of the system became more and more apparent as the mine grew, and tentative plans had been discussed from time to time, but the disadvantages of any new method were more apparent than the advantages, and nothing was done.

Attention was again and more forcibly drawn to the subject when the sudden loss of the Holbrook shaft in July, 1906, diverted more ore to the Czar and Spray than they could well handle. Tramming costs were raised by the increase in the distance trammed, and by congestion at junction points or at the stations. This resulted in heavy expense, which lasted six months until the new 559-ft. shaft could be finished and equipped.

Plans were begun on a general system for tramming and hoisting, which were later extended to include a power plant, and thus began a centralization of the surface plant and work, which has only recently been completed by the construction of a central timber-framing mill and a drill- and tool-sharpening shop.

Formerly, ore was hoisted through five shafts. Beginning with the oldest and shallowest, and passing to the southeast, they were as follows:

Shaft	Depth	Production of Dry Ore per Month, 1906
Czar.....	417 ft. to 400 level	7,814
Holbrook.....	559 ft. to 500 level	17,263 (1st 6 mo. only)
Spray.....	1,042 ft. to 900 level	18,530
Gardner.....	1,032 ft. to 1,000 level	7,560
Lowell.....	1,215 ft. to 1,200 level	4,327
Sacramento.....	928 ft. to 1,000 level
Total for year.....	564,154

The Holbrook has since been sunk to the 600 level and the Lowell to the 1,600.

Of these, the Holbrook, Spray, and Lowell shafts were either in the ore zone or so near stoped areas that there was continual movement in them, and some little danger of a repetition of the Holbrook collapse.

Plumbings were made regularly to ascertain the extent and rapidity of movement in the new Holbrook shaft. During the first two years, the collar moved to the northeast at the rate of an inch per month. It is now 6 ft. 2½ in. N. 45° E. of the stationary part below the third level.

Although the Spray shaft is in rather solid limestone, a crack developed, which passed through the shaft and split the engine foundation.

It may be noted in passing, that the engine foundations at these shafts now consist of two massive blocks of concrete, reinforced with old rails, each heavy enough in itself to hold the engine securely, and resist the rope pull. The ground beneath was leveled off by a layer of concrete, but the surface was separated from the foundation block, so that there could be no bond which would tear the concrete block apart in case of future movement. The fissure at the Spray shaft has since widened, but with no more effect than to tilt the foundation backward, engine and all. It runs smoothly on a slope of $\frac{3}{8}$ in. in 1 ft.

The Czar, although in rather heavy ground, was stable, and the Gardner is outside the ore zone and will probably remain true.

Each shaft was an independent unit, with its own boiler plant, etc. The equipment was miscellaneous, having been added to from time to time, and some of it was very old. The operation of the several plants was inefficient, from the number and small size of the units. The use of power was unimportant aside from that for hoisting.

New Plan Included Electric Haulage and Central Hoisting

It was proposed to equip a shaft, located in some central and safe position, for economical hoisting, where the ore might be mixed and sampled. The ore was to be drawn to the shaft by electric locomotives.

The advantages to be expected may be stated as follows:

1. The hand tram would be cut down to an average of 300 ft. or less.
2. Repairs would be reduced by diminishing the number and length of tramping drifts in bad ground. Access to stopes may be by circuitous routes to avoid heavy ground, and need not be so well maintained as is necessary when large quantities of ore must be drawn through them, the main-haulage drifts being placed beneath the ore in solid ground.
3. Fewer trammers would be required, and they could be distributed to avoid interference. There was often great congestion near the shafts, even when mining less ore than in recent years.
4. Hoisting would be cheapened by the use of skips, a more efficient engine, and a better method of transporting the ore to the railroad cars.
5. The unstable shafts would not be so essential to continuous production.
6. Shaft and cage repairs would be diminished, particularly in the moving shafts, by slowing down the hoisting speed, as could be done if only men, timber, and waste were hoisted.
7. Miscellaneous work about the shafts and auxiliary equipment might be cut down.
8. Power distribution from a central plant would be simplified and the investment for hoisting machinery reduced.
9. The ore could be thoroughly mixed and accurately sampled before shipment.

There were also certain disadvantages, aside from the cost of the improvements:

1. Haulage drifts must be driven and maintained without abandoning a corresponding length of hand-tramming drifts.

2. The ore would be transported underground a greater distance than before. A part of the saving in the length of the hand tram would be lost in the added operation of power haulage.

3. The old shafts must still be maintained for men, timbers, and waste rock, while the main shaft is added to those in use. (The new State law restricting the time of underground labor to 8 hr. from collar to collar, has made it important to have shafts close to the stopes and active workings.)

The advantages were believed to outweigh the disadvantages, and the construction was authorized.

Electric Haulage on Alternate Levels Only

The cost of equipping each level for electric haulage was prohibitive. Instead, each alternate level was chosen, beginning with the 400, making seven in all to the 1,600. By collecting the ore from 15 producing levels to seven haulage levels, the traffic density is increased, although still far below that necessary for really cheap transportation.

By referring again to the haulage map, it will be seen that the 400 level draws ore from the whole of the shallow western end of the mine. It begins in the solid ground under the stopes, passes out to one side of the ore zone and then to the hoisting shaft. It was possible to select old drifts in stable ground for most of the main line. The only new drift of any length was from the Gardner to the Sacramento. Ore is dropped from the 100, 200, and 300 to this level, through transfer chutes, under or near the stopes. In some cases, the haulage cars are collected by hand through short branch drifts to the stopes on the haulage level; in others, when the quantity warrants it, the trolley wire is carried to the stope chutes direct. Of the whole 5,600 ft. of main-line trolley drift on the 400, not more than 1,500 ft. was timbered, and of this only 450 ft. is particularly heavy.

In a similar manner, the 600 level was driven through from the Sacramento to the Gardner, old drifts were used to the Spray shaft, which were extended by new drifts, to reach the ore as it dipped below the 400 level. It underlaps the 400 haulage for some distance. Although this drift passed through the ore zone between the Gardner and Spray, some of the old drifts were in good ground, and the proportion of heavy ground to sound is not much greater than on the 400.

The same plan was carried out for the other levels, each one collecting ore from an area deeper, and farther to the south and east, than that served by the level above. Not very many new drifts were driven par-

ticularly for this purpose, and the percentage of main drifts which require repairs is small.

A trolley line on the 200 level to the Uncle Sam shaft was not contemplated in the beginning, but was added afterward. The ore from that shaft and the Southwest country is hauled to a transfer chute near the Holbrook shaft, where it is dropped to the 400. Extending the 400 level to the Uncle Sam to avoid re-handling may be justified at some later date.

To begin with, 7,000 ft. of new drift was driven especially for power haulage, but other drifts have been driven since, and others equipped for the purpose, until at this date, 10.9 miles of track and trolley line are in use or are standing ready. The connecting drifts on the 1,200 and 1,400 levels were warranted as prospects, and many others now used for haulage were driven to find or develop ore. Probably not more than one-quarter of the whole should be charged to the haulage system.

Track is 20-in. Gage

Twenty-five pound rails on 20-in. gage are used for the track, with 4 by 6 by 42 in. ties. They have been heavy enough except in a few cases.

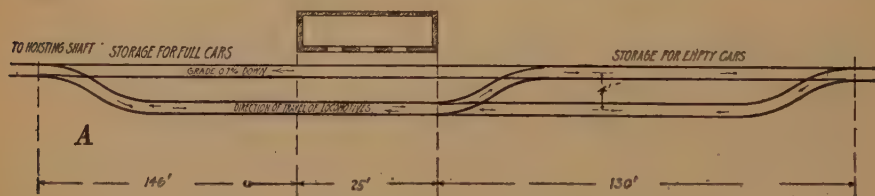


FIG. 2.—TRACK LAYOUT AT LOADING STATION.

The maximum concentrated load is on the 400 level. Locomotives are there used weighing 14,000 lb. The weight of a loaded car rarely exceeds 6,000 lb., although it may be more when loaded with massive sulphide ore. Wet ground on this level caused very heavy track repairs; even when slag was used for ballast, it was churned out from under the ties. To improve this condition, 4,200 ft. of second-hand 40-lb. rails were laid to replace lighter rails. The stiffer rails distributed the load better, and repairs were much reduced.

As a rule, the train crew does not load the ore. The motorman brings empties to the stope or transfer chute, and takes out the full cars standing there, or returns when the others are filled.

In the larger stations, sufficient double track is laid for tail room to hold 20 cars. A standard track layout is shown in Fig. 2. Only one switch, at A, is thrown by hand; spring switches direct the locomotives without further attention. At unimportant chutes, less elaborate sidings

for car storage are used, or the trains may be made up in the stope drift or on the main line.

At the Sacramento shaft, the tracks are looped to avoid switching.

Standard switches on curves of 40 to 80 ft. radius are used. Curves of 25-ft. radius are permitted only in exceptional cases.

Direct Current at 250 Volts Used Underground

Direct current is carried into the mine at 250 volts, through the Sacramento, Gardner, and Holbrook shafts. The feeder cables are cambric insulated, lead covered, and wire armored. The return cables have weather-proof insulation only. The voltage on the west end has been low, for lack of capacity in the power cables. A transformer set has been purchased to be set up at the Holbrook shaft to serve as a booster, but it is not yet in service.

The current underground is carried on 00 B & S American standard grooved trolley wire. The rails are bonded with No. 12064 Type E O. B. rail bonds, length 32 in. center to center, capacity B & S. No. 0, diameter of compression terminals $\frac{5}{8}$ in.

Trolley wires are now required by law to be 7 ft. above the track. They are hung from the timbers in timbered drifts, or from 4 by 6 in. cross bars held by vertical bolts wedged into drill holes, where the ground is hard. The old lines were sometimes hung lower, and in such cases were protected by 1 by 6 in. battens placed parallel with the wire, 8 in. apart and projecting 2 in. below the wire. On straight track, the wire is supported every 20 to 25 ft., as very little sag is permissible. Occasionally, the wires must be protected from acid water dripping from the roof. There is also more or less trouble with bonds in wet drifts.

There have been five fatal accidents in which the trolley current has been involved:

No. 1, September, 1908. Pipe-fitter at work on air line above wire. Received shock and fell to track 7 or 8 ft. below. Uncertain whether shock or fall was fatal.

No. 2, December, 1908. Miner repairing drift stood on a truck and fell off. Neck was broken. Thought that shock from trolley wire caused the fall.

No. 3, November, 1909. Loader carried chute bar when switching cars and struck wire beyond protection.

No. 4, May, 1912. Miner climbed over couplings of standing train. Tilted steel on shoulder and touched trolley wire.

No. 5, July, 1914. Miner repairing drift. Ground fell, carrying down trolley wire and pinned him down with wire against his back. May have been killed by the fall. Current was left on trolley wire to light the drift, contrary to rules.

The first two cases were doubtful, as the shocks may not have been

sufficiently severe in themselves to cause death. The danger of a current of 250 volts was not realized at once, as it is the lowest in common use, and other mines have not found it so. Other men also in this mine have received shocks without injury. These fatalities were supposed to be due to unusual susceptibility on the part of the individuals. It now appears most probable that when shoes are wet, as often happens, with acidulated water, and there is moisture on the hands or body, an unusually good contact may be made, with fatal results.

When this condition was recognized, special precautions were taken to prevent accidental contacts. The wires are high enough to avoid being touched ordinarily. Any reasonable increase in height would not entirely prevent contact when carrying tools, etc., on the shoulder, which is also contrary to rules.

The greatest danger is in repairing drifts or loading ore. No repairs are now permitted in the vicinity of live wires, and the system has been divided to facilitate cutting out the current in sections under repairs, without interfering with ore haulage. At loading stations, the wires were at first protected by boards on each side of the wire. The current is now switched off the wire on each side of the chutes for such a distance that the loader cannot receive a shock while loading, or when switching cars, even in case he should carry a chute bar with him when doing so. When the loading station is on a main line, the current is carried past on an insulated cable, and is only turned on the trolley wire when the train passes and after warning the loader. Current in the wire is indicated by a light.

There are a number of loading stations on each level, and as it takes as long or longer to load, than to make the trip to the shaft, dump, and return, the locomotive usually draws from more than one station. On the 400 level, where the ore is most difficult to handle, and there are many loading stations, they are connected by wire to a signal box containing an annunciator similar to that used in hotels. The loader can thus signal to the motorman when his train is nearly full, and save a trip to a station where the train is not loaded.

Red lights are used in a crude hand-operated block system when there is danger of interference with timber or waste handling, or when there is a second locomotive at work on the same level.

Colored lights are also used to indicate the position in which switches are set, or the location of a place of refuge from passing trains.

Factors Affecting the Choice of Cars

So many of the old drifts were to be used when the system was planned, that it seemed impossible to use large cars.

The first cars bought held 21 cu. ft. They had gable bottoms, with hinged sides, and were equipped with M. C. B. Midget couplers, spring-

draft rigging and roller-bearing axles. The latches were to be sprung for dumping at the shaft bins, by a block between the rails.

This type of car was not satisfactory. A large proportion of the ore is too sticky to slide out of the cars, even when the doors are held open. It was customary on the 400 for two men to rock them from side to side on the rails, in order to clear them of ore. The frictional resistance to dumping increases as the size diminishes.

Another disadvantage was their tendency to leak fine ore, particularly if wet.

After using them for some months, a few rocker dump cars were designed and built, of about the same capacity. Spring-draft rigging, automatic couplers and roller bearings were used, as they had been found advantageous. The rollers in the first bearings were solid. For the new cars, an axle was designed to use Hyatt rollers, with special care to make them dirt-proof. They dumped freely, and were otherwise satisfactory.

Since it was necessary to secure a complete new equipment of cars, the question of size was re-opened, and a car of 33 cu. ft. capacity was adopted.

The cost per cubic-foot capacity decreases as cars increase in size. They should cost less for repairs also, in proportion to the ore carried, as the details of latches, etc., are heavier and stronger. Larger net loads can be carried in large cars, as the weight does not increase in proportion to the capacity. The cost of loading and particularly unloading is generally less. For these reasons, the largest car possible should be used.

The size of the cars in this case was limited by the cost of enlarging the drifts, and by the weight which can be conveniently handled by one man.

A large sum of money was spent in widening drifts for the 33-ft. cars. A car enough larger to be of appreciable benefit would have incurred prohibitive expense in widening the $4\frac{1}{2}$ miles of drift then in use, and an added expense for all haulage drifts to be driven in the future.

At important loading stations, the track is laid on a grade to favor the loader in assembling cars. In many cases, the quantity produced will not warrant special sidings, and the cars must be loaded and pushed by hand to the point of assemblage. The 33-cu. ft. car weighs 1,700 lb. Loaded with dry, oxidized ore it weighs about 4,250 lb. When loaded with sulphide, it may weigh as much as 7,500 lb. This car with roller-bearing axles can be readily moved by one man at all transfer chutes, and through branch drifts in most cases, unless the grades are unfavorable. A larger car would quite frequently need two men. The use of cars larger than one man can move would increase the expense of switching by hand, until a car twice as large is used, while that of loading would

be practically doubled, since there is only room for one man to work conveniently at a chute.

The rolling-dump type was selected for its ease of dumping sticky ore, and simplicity of construction. Tipples were not seriously considered, as seven would have been required.

The advantages of such refinements as spring-draft rigging and automatic couplers may be open to argument, but a few fingers have probably been saved by the latter.

The roller-bearing axles (Fig. 3) have been an unqualified success. They are made in two sizes, and are of as great advantage on small cars as on large. The axles on the motor cars are filled with grease once a month; on the hand or mule cars once in two months. It is probable that this time is shorter than necessary, and experiments are being made to determine how long the charge will last. There has been an economy in lubricants accompanied by a very low depreciation charge. A few of the first Hyatt bearing axles, in use since January, 1909, have begun to fail. The cage goes first, which allows the rollers to become displaced and injured. These axles are $1\frac{3}{4}$ in. in diameter and are used on 22-cu. ft. cars. They are rather light for the service. The larger cars have $2\frac{3}{4}$ -in. axles, which appear to be amply large for the load, and a longer life is expected.

The friction load is very light. Unfortunately, the average mileage per car cannot be known.

TABLE I.—*Wear of Car Wheels, Axles and Roller Bearings*

Size of Car, Cubic Feet	Wheels				Roller Bearings			Axles	
	Date Put in Service	Material	Original Diameter, Inches	Wear in Diameter, Inches	Make	Original Diameter, Inches	Wear in Diameter, Inches	Original Diameter, Inches	Wear in Diameter, Inches
22	Jan., 1909	Manganese steel.	14	0.44	Hyatt	$\frac{9}{16}$	0.053	$1\frac{3}{4}$	0.013
33	June, 1910	Cast iron	14	0.41	Hyatt	$1\frac{5}{16}$	0.0075	$2\frac{3}{4}$	0.026
33	June, 1910	Cast steel	14	0.375	Hyatt	$2\frac{7}{32}$	0.008	$2\frac{3}{4}$	0.04
33	June, 1910	Cast iron	14	0.34	Solid	$2\frac{7}{32}$	0.023	$2\frac{3}{4}$	0.062

The roller bearings on which the above tests were made were manufactured either five or six years ago. The Hyatt Roller Bearing Co. advises that the rollers are now made of chrome-nickel alloy steel, which has a very much higher elastic limit than the steel used in bearings formerly.

The limit of accuracy in finishing is now approximately 0.002 in. instead of 0.005 in. It is very probable that better results would be obtained with the rollers as they are now made.

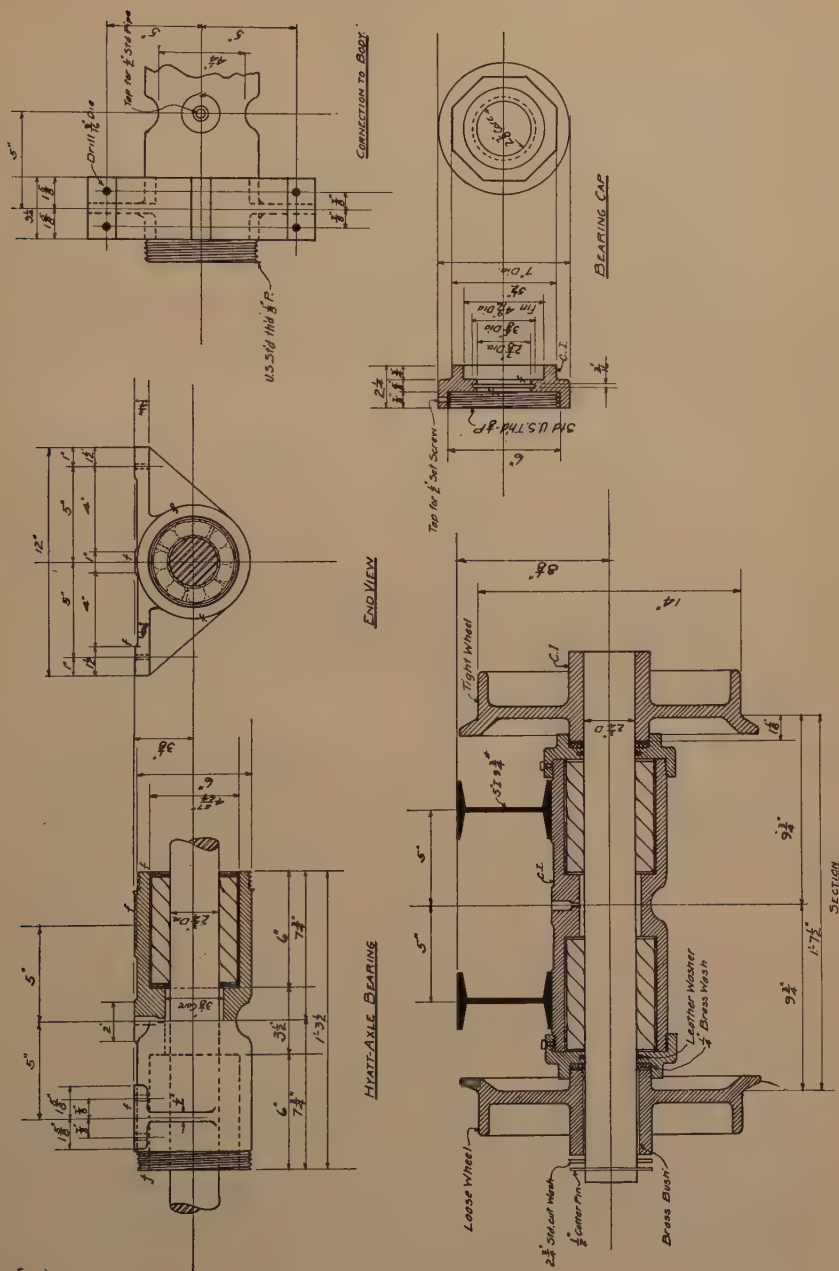


FIG. 3.—AXLE BEARING FOR 33 CU. FT. CAR.

Several Types of Cars Used

Several types of cars have been used, and a summary of their characteristics may be of interest.

The original car had a capacity of 12 cu. ft., and was of an ordinary type, dumping forward and also, by swivelling, to the side. It is not worth serious consideration, as it was both light and weak. The older shafts were small, and a car of greater capacity of this construction could not be put on the cages.

Two cars of 16- and 20-ft. capacity each (Fig. 4), dumping only to the side, were recently designed after a car developed by the Old Dominion Copper Mining & Smelting Co. The 16-ft. car will go on the cages in the smaller shafts. The 20-ft. car can only be used in the Gardner,

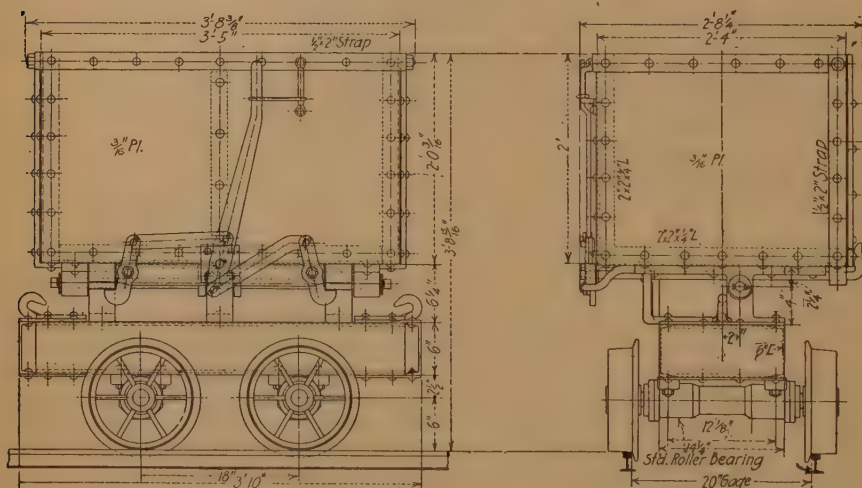


FIG. 4.—16-CU. FT. SQUARE SIDE-DUMP CAR.

Sacramento, Spray, and the newer prospect shafts. These cars have great capacity for their overall dimensions, are of heavy construction, and can be dumped without uncoupling when used in trains. This is an important advantage, as they are frequently hauled in short trains by mules or motor to transfer chutes, or in carrying waste from development work. The disadvantage is that they can only dump to one side, and care must be taken to see that they come to the dumping place faced the proper way. It is proposed to replace the 12-ft. cars with these as fast as they wear out. It is unfortunate that two sizes need to be used, but it is believed that the added capacity of the large car makes it worth while.

Rolling-dump cars of extremely simple construction were designed for use at the North Star Mines, in Grass Valley, Cal., and were introduced

at the Copper Queen for surface work. Afterward, they were taken underground, and have proved satisfactory for hand and mule tramming. They are cars reduced to their lowest terms, consisting as they do of a pair of axles with wheels, two beams of "I" or channel section tied together at the ends by two cast-steel supports for the body, which also form bumpers and attachments for connecting in trains. In small sizes, they may be locked by a pin through trunnion and support, if necessary. When properly balanced, they dump with great ease, and do well with sticky ore. As the body forms a tight box, ore is not spilled, even if it is a thin mud. As they dump more easily when well filled, the tendency is to heap them up rather than load them to only a part of their capacity.

They are bulky for their capacity, however, and while this is not important when used exclusively on levels, it is an objection for cage work. They are also high and not so convenient to shovel into. In order to preclude a multiplicity of types, it is probable that the square-bodied side-dump car will eventually supersede these also, as they can only pass through the newest shafts.

The 33-ft. car of this type, which is used for haulage, is too large for general hand or mule tramming, and has most of the advantages and disadvantages of a small car of the same type. It dumps muddy ore more readily than any car which has been tried, but if the ore does not all slide out together, what remains is in the bottom and it is hard to hold in position to dump. The point of support is placed below the center of gravity in order to dump easily, and a lock on the bottom prevents its over-turning in transit. A larger-sized rolling-dump car is not to be recommended. The force necessary to revolve a weight of 2 to 3 tons may be excessive.

In larger sizes also, the objections to the saddle-back car disappear. They dump ore well when they hold 75 to 100 cu. ft. The doors are the only moving parts, the ore is merely permitted to fall out, and, if it does, nothing better can be desired.

Three to 7-Ton Electric Locomotives in Use

There was difficulty at first in securing standard locomotives small enough to go through the drifts; 3-ton Goodman locomotives were selected, their type No. 1600-K. They were small and compact, and were driven by a single motor of 20 to 25 hp. on one-hour rating. Since, under ordinary conditions, the train crew does no loading, the locomotives were in continuous motion. This gave the motors no time to cool off, armatures burned out, and electric repairs in general were very heavy. About this time, it was decided to use larger cars, the drifts were widened and larger locomotives could be used. Two 6-ton locomotives (6 N-0-2) were then purchased. They were driven by two 8-A Goodman motors of 22 hp. each on one-hour rating, and have been satisfactory.

The haul on the 400 level is long, with adverse grades for a third of the distance. A still larger locomotive was thought desirable for this level, and two 7-ton machines were designed and built in the mine shop. They are extremely small for their weight and power. The frame is of cast steel. The two Westinghouse No. 79 mining motors, of 30 hp. each on one-hour rating, were taken second-hand from stock. In the design, everything was sacrificed to compactness and power, and it is not to be recommended for general purposes. The unusually rugged electric equipment has fulfilled every expectation in the elimination of delays and repairs. The tire expense is rather heavy, since the clearance below gears is so small, the tires can only be worn down 1 in. in diameter before replacement.

The small locomotives replaced by the heavy machines have been used to good advantage on light work, either on levels where the ore production is not heavy, or in hauling waste or timber. They were well constructed, serviceable machines when used with the load for which they were designed. Since then, repairs have been general and not confined to any particular feature. Gears and tires make up most of the operating cost. The gears are cut out of special steel blanks on a milling machine at the mine shops. The tires are of forged steel shrunk on a cast-iron center. The treads are re-faced when necessary, and are worn down to a thickness of $\frac{1}{4}$ -in. at times. Manganese-steel tires were tried, but were found to wear very rapidly.

Odometers have only recently been attached; accurate mileage figures therefore cannot be given.

In narrow drifts, it is sometimes impossible to turn the trolley pole, and many were broken in backing up. A trolley pole with a knuckle in the center, which is easier to turn and which can be turned anywhere, has reduced the breakage.

In selecting a locomotive, it should be seen that the motor equipment is large enough for the work to be done; it will save endless trouble, delays and expense.

Power-Rating of Locomotives per Ton on Wheels

Size Locomotive, Tons	One-hour Rating, Horsepower	Horsepower per Ton
3	20	6.6 to 8.3
6	44	7.3
7	60	8.6

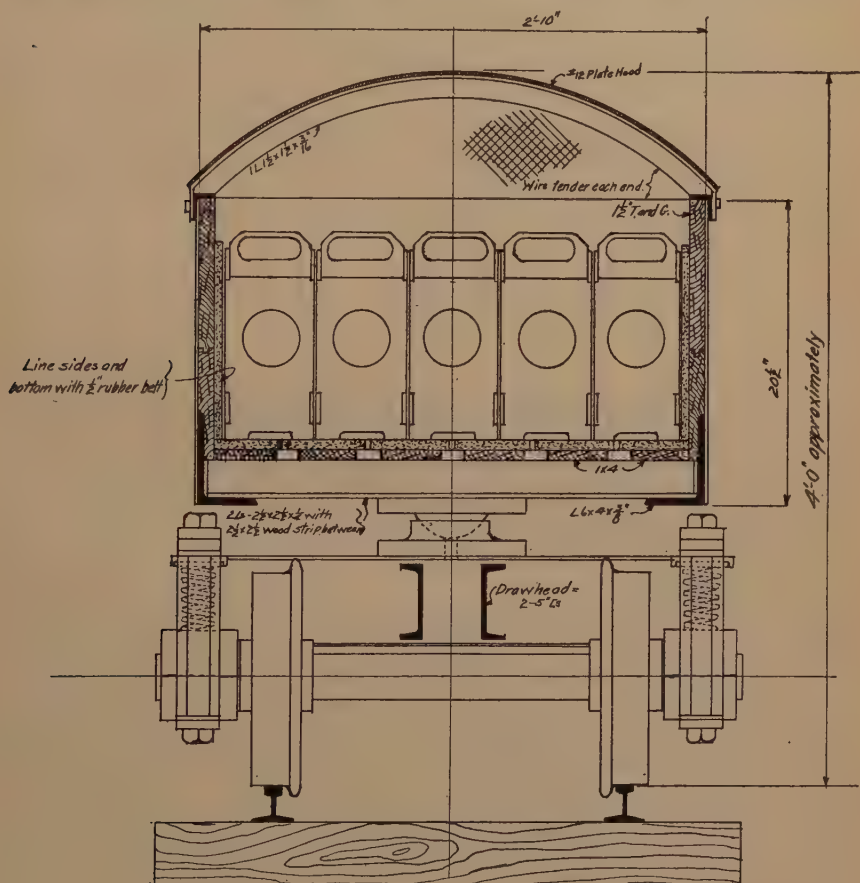
Storage-Battery Locomotive Tried

One of the 3-ton locomotives was used for a time in 1910 and 1911 with an Edison storage battery. The results of the running test were presented in a paper by Charles Legrand.² The battery was composed of 150 A-6 cells carried in two separate trailing cars.

² Electric Traction in Mines, *Trans.*, xlviii, 295 (1914).

Moisture from the air accumulated on the cells, to which was added a small proportion of electrolyte as a result of spillage or gassing while being charged, and a number of cells were lost by short circuiting.

In 14 months, 21 cells were burned out. The cells at the ends of the cars were those usually lost. The whole battery was then returned to the factory, and replaced by another with different crating to provide



Cross-Section of second Car

FIG. 5.—UNDERGROUND MOTOR STORAGE-BATTERY CAR.

better insulation. A new battery car provided with springs (Fig. 5) was built large enough to contain all the cells. It weighed 2,400 lb., and the battery 3,000 lb. The cells were occasionally wiped dry, painted and coated with vaseline. The nickel walls of but two cells were burned through in $7\frac{1}{2}$ months' service.

If the batteries could have been kept perfectly dry, or if the voltage

had been lower, there should have been little or no difficulty from short circuiting.

When idle and fully charged, the voltage was 213; after travelling about 8 miles, one-half under load, the voltage dropped to about 200. When the voltage dropped still lower, the speed of the locomotive decreased and the work went slowly.

Ore was drawn during the day and the battery was charged at night, by current from a 20-hp. motor-generator set taking power from the alternating-current circuit. It was automatically cut out when charging was complete.

The production from the area served by this locomotive gradually increased until its capacity was exceeded. It was then thought better to lay a wire than to get another battery, and the use of the storage battery was discontinued. The cost of the locomotive was approximately \$1,300, that of the battery about \$3,000.

The operating costs were reasonable in spite of light rails and sharp curves. No information on normal depreciation can be given.

Favorable conditions for motor and battery are opposed. In order to protect the battery, the motor must not be too large. And yet it has been found that repairs are reduced by making the motor powerful enough to slip the wheels of the locomotive readily. The added weight of the battery gives it good traction, and there is always a temptation to pile on the load. By incorporating the battery with the locomotive, they may be better suited to each other and the work to be done.

The capacity is not very great. The cost of battery, charging station, etc., is to be balanced against that of trolley wire and rail bonding. The charge for power is so small that the greater consumption of power-house current is of no particular consequence.

The battery is now set up in the power house to excite the fields of the alternating-current generators when starting. It replaced a small steam-generator set, and has been very useful.

Several Designs for Slope and Transfer Chutes

When the stopes are on a haulage level, only the regular stope chutes are used, and the cars are brought to them by the locomotives, if the tonnage is of sufficient importance, or by hand from some nearby point where they may be re-assembled into a train to be hauled to the shaft. If the ore comes from the level above, it is dropped through a transfer chute to the haulage level. This may be an old stope chute or it may be a special raise into which the ore from a considerable area is collected.

For the larger bodies, a chute is usually provided close to the stopes, even if a special drift must be brought in from the main line. In a top

slice, the main chute is brought up to the top floor and the ore is trammed from the working faces to the haulage chute direct. In the Dividend slice, the ore being too soft for raises to be maintained through it, they are placed in the country rock outside, and the ore will be trammed a longer distance on the working floor of the slice, instead of to a central raise within the ore.

In the Howell slice, the ore mined above the 600 is dropped to the 1,000, as it is economically impossible to maintain a trolley drift through the ore zone on the 800. In this case, the ore is trammed again on the 900, as there is not sufficient to justify a special raise clear through.

Table II.—Loading Cost for Levels in 1914

Level	Cents per Ton
400.....	6.4
600.....	2.4
800.....	3.4
1,000.....	3.5
1,200.....	1.8
1,400.....	2.8
1,600.....	3.2

The 400-level costs for loading are highest, since the greatest proportion of clay ore originates in the area tributary to that level. The ore in the Dividend stopes is a very tenacious clay, and a cost of 10c. per ton for loading has not been uncommon or excessive. The difficulties of loading have directed particular attention to the development of a better chute than that formerly employed.

A rectangular vertical chute of small section, with a bottom sloping down to a door in the front, is the worst possible for sticky ores. The small section increases side friction and favors arching. The weight of the column of ore packs it tightly into the wedge-shaped section at the bottom, particularly when it falls as a mass after hanging up. Only a part of its weight acts as a force to push it horizontally through the door. By shortening and widening the column of ore, the weight per square foot on the bottom layer is lessened, and there is less possibility of arching. By putting the door underneath instead of at the side, the full weight of the ore acts to force it through. These principles were used in developing a chute which has been found to be of great advantage. The doors have been put underneath the bin in only one case, as it is seldom necessary.

From the sketch of a standard transfer chute (Fig. 6), it will be seen that the storage is all at the bottom. The ore falls through the chimney and strikes a baffle, its downward velocity is checked and it falls lightly on the heap below. The pile of ore is not allowed to extend up into the raise, which serves only as a passage way. The storage space is long enough to let it spread into a loose pile, there is little tendency to

pack, and it cannot readily arch. Below, there are several doors. If, when loading, the ore hangs up over one door, the car is moved to the next, which will break the arch, or if that door does not, to the third. The bin itself is usually placed in solid ground and requires no timber except in the bottom. The narrow chute above is only lined in heavy ground, and then usually with concrete. Its small diameter, from 3 to $3\frac{1}{2}$ ft., brings this expense within reason.

Seven chutes have been lined with concrete. One which has been in use for 20 months, needs re-lining, due to wear by sharp ore. This

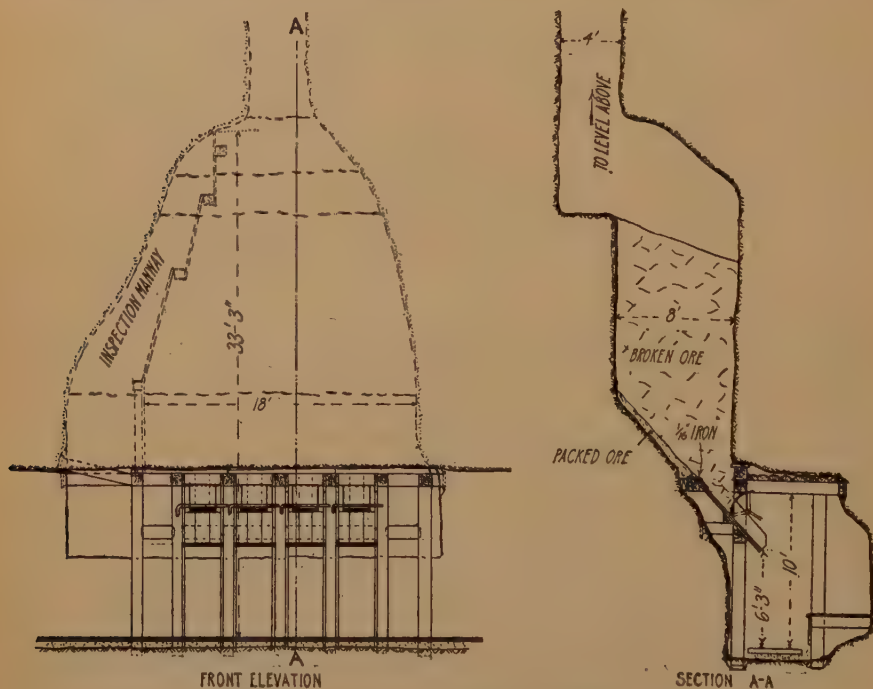


FIG. 6.—STANDARD HAULAGE CHUTE. 90 TONS CAPACITY.

chute would have required heavy repairs if timbered, both to provide against outside pressure and to replace the lining. In re-lining, a thicker wall of concrete will be used. Other chutes are only slightly worn.

An elaborate storage bin, shown in Fig. 7, was built on the 400 level to serve a large aluminous orebody which has been developed, but not yet mined. It was described by Joseph P. Hodgson, Mine Superintendent of the Copper Queen mines, in a paper read at the Salt Lake meeting of the Institute.³ A large tonnage of ore is tributary to it which is so sticky that extreme measures were considered necessary.

³ Mining Methods at the Copper Queen Mines, *Trans.*, xlix, 316 (1914).

This is the case in which the doors are put underneath the center of the pile of ore. There is an opening from one end to allow the loader to climb up on the pile, if necessary, for inspection, or to work the ore down. A $\frac{3}{4}$ -in. pipe is laid in the concrete lining, to serve as a speaking tube between the loader and the trammer above.

The ore from this stope had formerly been dropped from the 200 to the 400 through a standard, timbered chute. No attempt was made

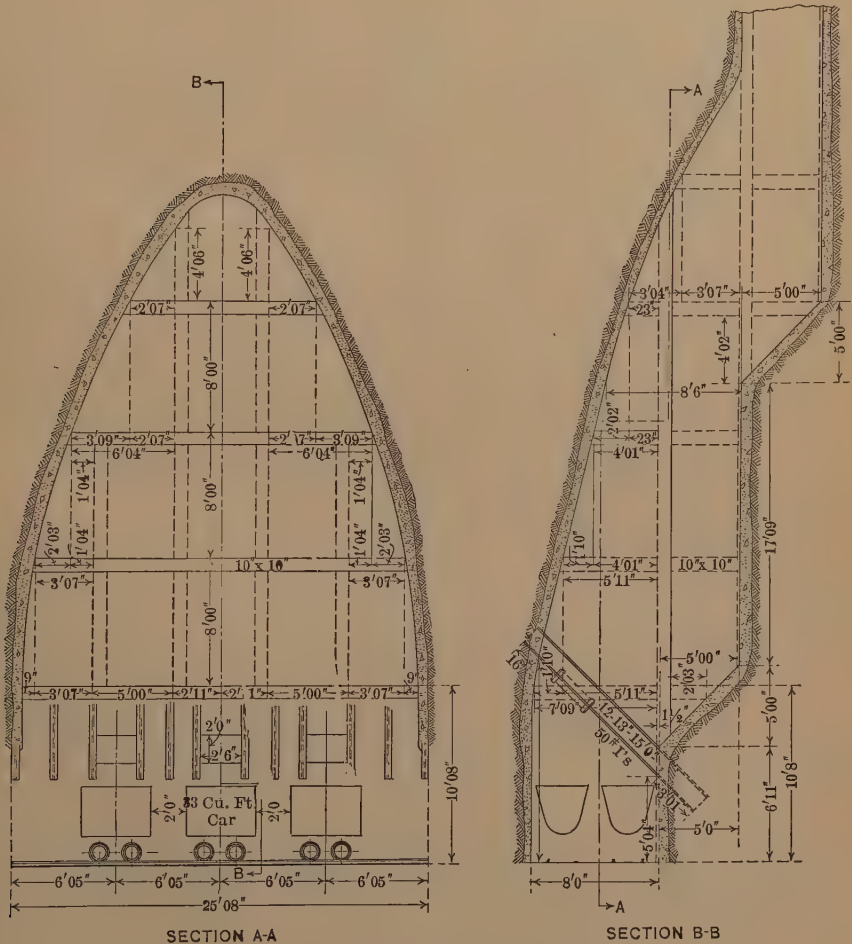


FIG. 7.—CZAR ORE POCKET.

to use the chute for storage, and the door at the 400 was kept open by the loader, but other men were required above to keep the chute clear and the ore moving.

It was found that the whole capacity of the storage bin could not be used, the weight of the ore packing it too hard; but if the quantity

does not exceed 50 or 60 tons, the loading is rapid, and there has been no accumulation of ore in the concrete chimney at any time. In rebuilding it, the storage chamber would not be made so high.

The timbered chute shown in Fig. 8 is sometimes used successfully for stopes. The half bulkheads partly support the ore, and it does not pack so badly in the bottom. It was used by the Cananea Consolidated Copper Co.

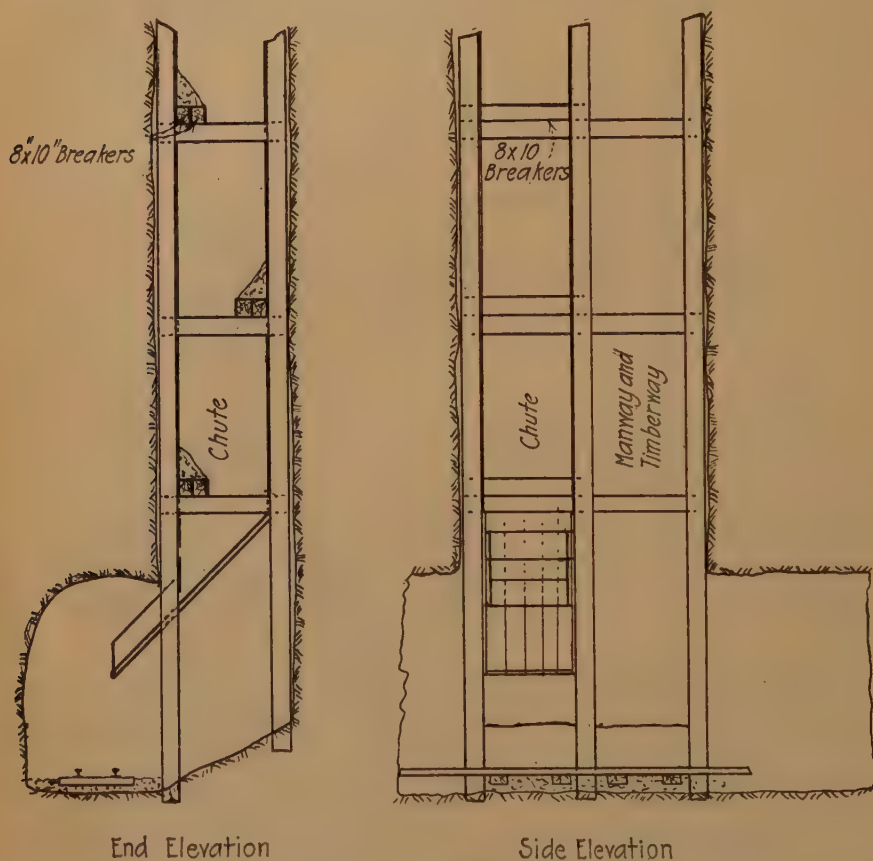


FIG. 8.—TIMBERED CHUTE.

Sacramento Selected for Hoisting Shaft

The greatest month's production prior to July, 1907, had been 55,347 dry tons, in January, 1906, or 1,910 tons per working day. It was not exceeded until March, 1912. The desired capacity of the plant was assumed to be 60,000 tons per month, or about 2,150 tons per working day, with a reasonable margin for emergencies. At that time, the mine

worked alternate Sundays, and three shifts were run in tramming and hoisting.

The Sacramento shaft had been sunk for general development purposes, and had been located within the porphyry intrusion, to avoid the ore zone and its possible movement. It was selected for the hoisting shaft since it was in a central position and convenient to the main line of the railroad.

It was 928 ft. deep to the 1,000 level, and had three compartments 4 ft. 6 in. by 5 ft., and a half compartment for air pipes, etc. Since then, it has been sunk to the 1,700 level and widened to five compartments, making two for skips, two for cages and a large one 5 ft. by 5 ft. for pipes and cables.

It was at once decided to use skips.

The size of the compartments restricted the cross-section of the skip to 4 ft. 2 in. by 3 ft. 3 in. inside. The effective capacity fixed upon was 60 cu. ft. It was thought that one of large capacity would be too long, and that the bottom layer of ore might be packed so hard by the fall into the skip and the weight of a high column of ore upon it, that it might stick. This has sometimes been the case with the size adopted. The average skip load in April, 1915, was 8,117 lb. of wet, or 7,459 lb. of dry ore. The moisture, 8.11 per cent., is the lowest in years, the average would be nearer 12 per cent.

There would be no difficulty in raising much more than the quantity of ore assumed, when it runs freely, even with small skips. When it does not, large skips are of no particular advantage.

The uncertain factor was the rate at which ore could be drawn from the storage bins, which are necessary when skips are used. The experience with bins had not been particularly successful. Attention was therefore concentrated upon providing facilities for rapid loading, and to give loaders the maximum time for work at the loading door.

The size of the skips, power and efficiency of the hoist and economy of rope wear are interdependent. Conditions favorable to one are usually opposed to efficiency in the other.

A larger skip requires a more powerful and expensive engine, the efficiency of which is less, due to less continuous hoisting for the assumed tonnage, and greater condensation. There might be a small saving in rope expense, as the skips would make fewer trips.

Maximum efficiency in ropes would be obtained by larger drums, or drums long enough to avoid two layers. Larger drums require larger cylinders and again an engine more costly and less efficient. There is no great advantage in the use of larger drums unless they are large enough to hold the rope in one layer.

The capacity of the skips, the proportions of the engine, and diameter of the ropes were arrived at by one of those compromises, which must be

so frequently made in balancing an advantage against its complementary disadvantage.

Reports of the steam economy of compound engines operating in South Africa had been received, and after thorough investigation by John Langton, Consulting Mechanical Engineer for the company, a duplex, tandem-compound, condensing, first-motion engine was purchased from the Nordberg Manufacturing Co. The conditions were not the most favorable for a compound condensing engine, as the depth of the center of gravity of production in 1906 was only 570 ft., but it was expected that the depth of hoisting would increase. The center of gravity of production is now 970 ft. below the collar.

The plant is driven by steam at 150 lb. power-house pressure, transmitted 619 ft. through a 5-in. main. The steam leaves the power house with 75° F. of superheat. The engine can lift an unbalanced load of 17,000 lb. Its speed is rated at 2,000 ft. per minute. The reels hold 2,100 ft. of 1½-in. rope in two layers. The cylinders are 18 in. and 28 in. in diameter by 48-in. stroke.

Most of the details are of standard Nordberg design, but there are three unusual features:

The operation of the auxiliaries.

The regulation of pressure on the low-pressure cylinders.

The condensing system.

Auxiliaries Actuated by Oil under 135 lb. Pressure.

When steam is used by the auxiliaries, which operate brakes, clutches, etc., it constitutes an important percentage of the total power used by the engine. The condensation is great and continuous, and since the controlling valves can have no lap, but must be set "line and line," it is impossible to avoid leakage as well. The steam consumption by auxiliaries in a simple engine running at capacity is rarely less than 14 per cent. of the total. If the engine runs intermittently, the proportion used by the auxiliaries is correspondingly greater. In a more efficient engine, the steam for auxiliaries remains the same and is also a greater proportion of the whole.

In this engine, throttle, clutches, brakes and reverse are actuated by oil under a pressure of 135 lb. and controlled by poppet valves. The oil discharged from the auxiliaries is pumped back into an accumulator by a triplex pump, with plungers $2\frac{7}{16}$ in. in diameter by 4-in. stroke, which are driven by eccentrics on the valve-gear shaft. When the accumulator plunger is raised to its highest position, the oil is bypassed to a reservoir. When hoisting from the upper levels, the pump is too small and a relay steam pump which cuts in when the accumulator plunger is below a

certain point runs most of the time. This method is entirely satisfactory, and the engine control is sensitive, positive, and efficient.

The operating levers are arranged in a standard manner (Fig. 9) first used on the Fraser & Chalmers 20 by 48 in. engine at the Spray shaft. All the large hoists are controlled by levers on this plan. There is a great advantage in having all engine levers in relatively the same posi-

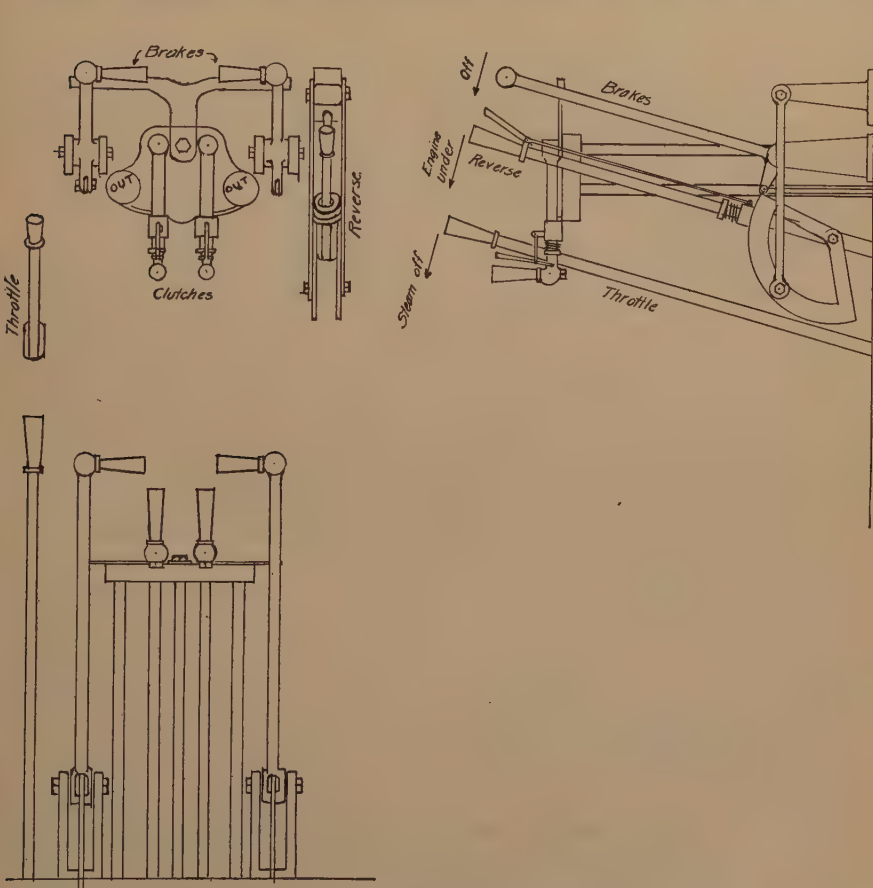


FIG. 9.—TYPICAL HOIST-LEVER OPERATING SYSTEM.

tion, acting by the same motion. Engineers can be transferred from one hoist to another without danger of confusion.

Pressure on Low-Pressure Cylinder Automatically Regulated

A large steam-jacketed receiver is placed between the high- and low-pressure cylinders, in which pressure is maintained at 20 lb. by an automatic valve, through which steam is bypassed from the high-pressure

main to the receiver, when the throttle is closed. Thus, the engine has pressure on all cylinders when starting, and picks up smoothly and quickly instead of needing a stroke or two to reach a working pressure in the low-pressure cylinders. When the throttle is opened, the bypass is shut off and the pressure in the receiver drops to that dependent on the load, speed, etc. When it is desired to hoist with great speed, quicker acceleration may be obtained by adjusting the reducing valve to raise the receiver pressure.

Condenser Operates under Severe Conditions

The engine has Corliss valves, with automatic cutoff regulated by a governor. Severe conditions are imposed on the condenser by the irregular use of steam. During the first two or three strokes, while the engine is starting the load, steam is taken nearly full stroke, while at the end of the trip the cutoff may be at $\frac{1}{10}$ stroke. In order to condense this sudden rush of steam, a large reservoir of cold water is provided in the "Rain Type" Nordberg condenser cylinder, into which the steam is exhausted, by a series of shallow flat trays filled with water. Constant streams of water are poured into them which overflow, rain down to the bottom of the cylinder and finally run off by gravity against the vacuum, to the cooling tower, from which the water is re-pumped. A 900-gal. rotary pump is used for circulating the water and a 19 by 18 in. vacuum pump, both belted to 20-hp. 220-volt alternating-current motors which run at constant speed. The temperature of the water at present (June, 1915), is 84° F. when entering and 96° F. when leaving the condenser. The barometer stands at approximately 25 in. The vacuum drops to $16\frac{1}{2}$ or 17 in. at the beginning of a trip, but rises to 20 or $20\frac{1}{2}$ in. The man engine also runs condensing and exhausts into the same system, which was increased in capacity by large pumps and a second condenser when the man engine was added.

Hoisting Speed 1,300 to 1,600 ft. per Minute

In order to insure continuity of operation under all conditions, the low-pressure cylinders are built to stand high-pressure steam. The receiver pressure can be built up to 150 lb. and the engine operated as a simple engine with cylinders 28 by 48 in. stroke. This makes it an engine of great power for emergency use, as for instance, hoisting unbalanced, but at a sacrifice of efficiency.

The engine governor is set for a rope speed of 1,300 ft. per minute when hoisting from the 1000 level or a level above, but by shifting the governor belt to another pulley, a speed of 1,600 ft. is used for greater depths. When production is increased, both speeds are raised. It is

preferred to run the hoist at as regular and slow a rate as will conveniently hoist the ore.

Round Rope on Grooved Drums Now Used

Round ropes, $1\frac{1}{4}$ in. in diameter, in two laps are used on reels, which are 7 ft. in diameter by 5 ft. face. The drums were made without grooving at first, as it was uncertain which ropes were best, a high-grade rope $1\frac{1}{8}$ in. in diameter or $1\frac{1}{4}$ in. rope of less tensile strength. The ropes began to cut in an irregular curve into the drums, which were grooved in April, 1914, to a depth of $\frac{5}{16}$ in., leaving the diameter of the drums at the bottom of the grooves $83\frac{3}{8}$ in.

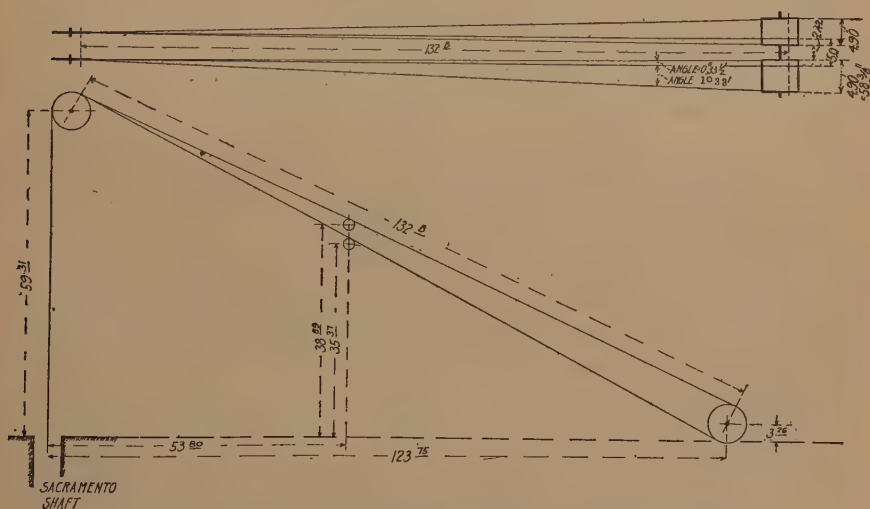


FIG. 10.—LEAD AND FLEET ANGLES, SACRAMENTO ROPES.

Fig. 10 shows the relative positions of drums and sheaves to shaft. The ground back of the shaft was cut out as far as practicable to get a reasonably long lead from the sheaves.

The fleet angles are unequal, and from the sheave line to the outside of the drum is $1^{\circ} 33'$. This would be considered excessive in certain districts, the copper country of Michigan for instance, but accepted practice in other places permits an angle of $1^{\circ} 55'$. The original attachments of the ropes were at the outside, or clutch side, of the drums. There was sufficient angularity in the pull from the sheave to draw the laps apart and leave openings into which the second layer of rope dropped. The winding was irregular and rough, and the wear excessive. Other sockets for rope attachment were cut on the inside of the drums, and the trouble ceased. The angular pull tends to hold the laps together in-

stead of separating them. When the drums were grooved, the ropes were again attached at the outside, since the active part of the rope would be nearest the sheave line, and the fleet angle smaller. There is heavy side friction on the rope when the last laps of the second layer are wound on. If attached at the inside of the drum, the grooving would carry it to the outside without so much friction as if winding against a rope. On the second layer, the pull of the rope would again tend to separate the laps and reduce the side friction of rope against rope. Soft-steel wedge-shaped fillers lift the rope out of the inside groove, high enough to wind back on the first layer.

No difference has been noticed in the wear of the under or over wind. If there is a reasonable distance between sheave and drum, there should be none.

The sheaves are 7 ft. in diameter, and are lined with leather. Wood lining broke down rapidly. Cast-steel idlers support the ropes between sheaves and drums to reduce slapping. They are cast thin and the flanges machined to make them still thinner and lighter.

The ropes are attached to the skips by thimbles and six Crosby rope clamps. Once every four weeks, 6 or 8 ft. is cut off the end of each rope to change the points of maximum wear.

The shaft is usually dry. In the wet season, a little acid water seeps in, but no corrosion has been discovered.

A few years ago, Mr. Duncan, Agent of the Cleveland-Cliffs Iron Co., kindly furnished rope records from the Lake shaft hoist, which has 7-ft. drums, 9-ft. sheaves and uses $3\frac{1}{2}$ -ton skips, but with only one layer of rope on the drum.

TABLE III.—*Data on Hoisting Ropes*

	Cleveland-Cliffs	Copper Queen
Diameter of drum, feet.....	7	7
Diameter of sheave, feet.....	9	7
Length of rope, feet.....	1,800	4,000 (2 ropes)
Diameter of rope, inches.....	$1\frac{1}{4}$	$1\frac{1}{4}$
Weight of skip, pounds.....	4,000	6,350
Weight of ore, pounds.....	7,000	7,400 (dry)
Weight of rope, maximum, pounds....	1,347	3,880
Ore and rock hoisted, tons.....	700,000	395,569 (dry)
Depth of average hoist, feet.....	550	970
Depth of maximum hoist, feet.....	550	1,550

The Copper Queen rope averages do not include those ropes which were obviously unsuited to the conditions.

The ratio of rope to drum diameter is 1:67.2. The experience of the Cleveland-Cliffs Iron Co. indicates that good service may be obtained with that ratio.

The rope cost on the Copper Queen ropes in the above comparison was 0.3 c. per dry ton. The ore contained about 11 per cent. moisture during the time covered by the records.

From the above comparison, the life of a rope is increased 75 per cent. by the use of one lap instead of two, and the cost for ropes at the Sacramento shaft would have been in consequence, 0.17 c. instead of 0.3 c. This would represent a saving in rope in an average year of about \$1,000. In order to obtain this saving, the cost of the engine would have been approximately \$15,000 greater, and it would be less efficient. A geared hoist would avoid some of these objections, but would introduce others.

The greatest tonnage, 549,324 dry tons, was hoisted by a pair of flattened strand ropes with Lang lay. The cost was 0.277 c. per dry ton. The increase in wearing surface provided by flattened strands is very noticeable. In winding on a smooth drum, the contact between two consecutive laps is made by a rubbing motion. In the ordinary lay, the outside wires which touch are parallel with the length of the rope at the point of contact, and the rough surfaces of the projecting strands bear against each other when the rope comes down against the last wrap. With the Lang lay, they are inclined to the line of the rope and each other, and the friction is reduced.

These conditions are of paramount importance when two layers are wound on a drum, since the bearing of the second lap is entirely rope against rope, and there is at the best a good deal of sliding of one rope on another. The external wear only need be considered, internal wear is of no consequence in comparison. Efficient lubrication on the outside of the rope is of great benefit.

This question is treated in some detail, because it was found very difficult to obtain information on the effect of small drums and two layers of rope on the cost of hoisting. It was of course recognized that the wear would be increased, but to what degree it would increase the cost could not be ascertained.

Underground Storage Bins Have Loading Hoppers

The storage bins at the haulage levels were at first made of standard design. The only attempt to avoid the difficulties formerly experienced with bins was to make them comparatively small to avoid pressure.

Attention was particularly directed toward giving the loader as much time as possible to poke the chute. This was done by putting measuring hoppers holding one skip load each below the bins, which could be loaded at any time while the skip was in motion and tripped into the skip when it reached the loading position. When hoisting in balance at the maximum rate expected, a skip per minute, and allowing 15 sec. for

tripping the hopper and ringing off the skip, each loader has $1\frac{3}{4}$ min. to load the hopper.

When starting the plant, it was found at once that the customary bin was not satisfactory. Not more than half the requisite speed could be attained. The 800-level bin was altered to one of the present type, but of small size, 40 tons capacity. As soon as it was found successful, the 400 and 600 bins were rebuilt but on a larger scale. Their capacity is 90 and 100 tons. The three lower bins are about the same size.

The bin as reconstructed consists of two parts, an upper chamber for storage and a lower part open in front which acts as a slide to the gates. The horizontal opening at the bottom of the storage chamber is 4 by 8 ft., large enough to prevent arching, but small enough to carry the weight of the ore in storage without permitting it to exert much pressure on the ore resting against the gate. The front, immediately below the contraction, is open for an added relief against pressure as the ore is free to take its natural slope.

This principle has since been used in a 260-ton bin for the 400 level, which is shown in Figs. 11 and 12, together with a standard wooden bin. In this case, the excavation was made circular and lined with concrete lightly reinforced. The conical bottom is armored with rails set in the concrete. It has been found necessary to protect the concrete against erosion when hard ore is stored, but only where the ore strikes on rebounding from the inclined plane at the bottom of the chimney.

It was not known at first how large the hole in the bottom of the cone should be. It was accordingly made 6 ft. in diameter, with the expectation of enlarging it if necessary. It has since been cut out to 7 ft. 4 in., which appears to be the correct proportion to protect the ore below from excessive weight and still prevent arching. This storage bin, although much larger than the others, handles muddy ore equally well.

As finally constructed, the large bin was built on the opposite side of the shaft from that shown in the drawing.

Arc doors 30 by 16 in. in size, are generally used, with good results. The slight extra labor of operation is balanced against the higher cost for installation and maintenance of power doors, which are only used at the large concrete bin at the 400 level.

Below the doors are the loading hoppers (Fig. 13) which hold 58 cu. ft. each, or one skip load. The flat counter-weighted bottom is hinged at the rear and held shut by tension bars on each side extending up to a shaft which is turned by hand levers. It dumps quickly and locks positively, as the tension bars are turned past the center of the shaft. The chute beneath, which carries the ore to the skip, is very steep, to prevent sticking.

The operation of loading is very rapid under ordinary conditions.

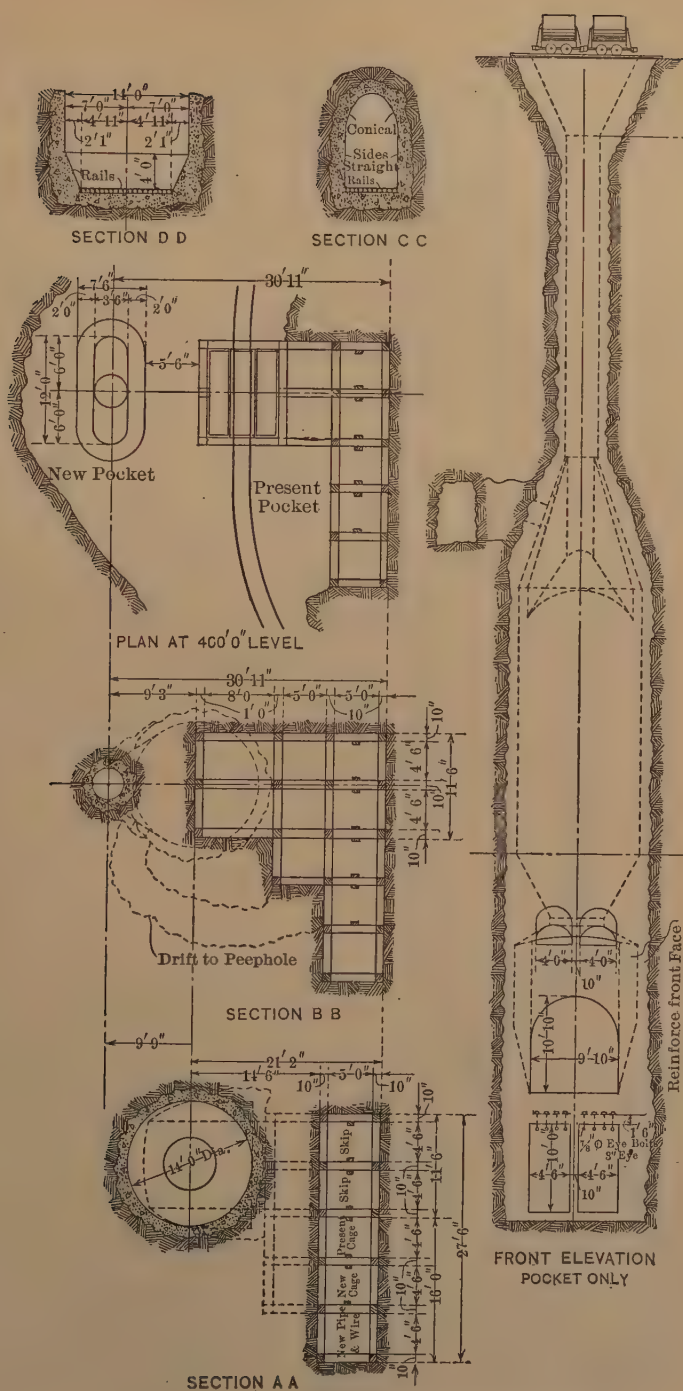


FIG. 11.—260-TON ORE POCKET, 400 LEVEL.

The time required to dump the ore into the skip and ring it off rarely exceeds 12 sec., and frequently not more than 10 sec.

The best day's record is 3,899 wet tons in three shifts, or 3,467 dry tons. An average has been maintained for several shifts at the rate of

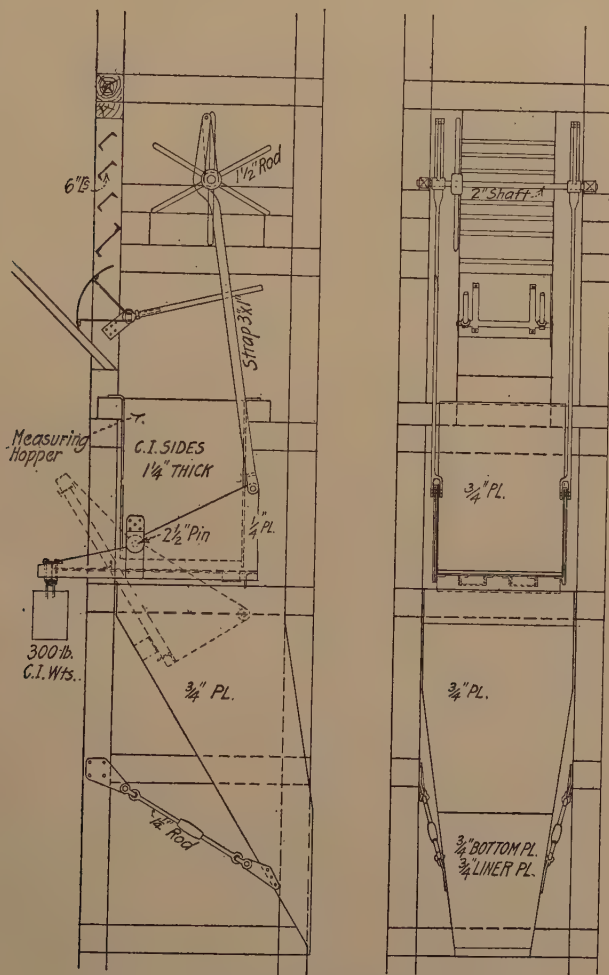


FIG. 13.—SACRAMENTO MEASURING HOPPER.

433 skips per shift. When the 30 min. for lunch and the time estimated for changing levels are deducted, it makes a hoisting rate of 1.1 skip per minute. Two loaders are required. When there is much muddy ore, or when hoisting very rapidly, a third man is sometimes used. There are still occasions when, for short periods, not more than 20 or 30 skips per hour can be hoisted.

The construction of skips and loading mechanism is heavy and strong. There have been few delays and light repairs.

There has been trouble with the spilling of ore while loading. The chute through which the ore reaches the skip is so steep that the opening is 40 in. high. Even if the skip is held only 3 in. below the bottom of the chute when spotted for loading, there is an opportunity for the ore to spread in falling, and for some of it to fall on each side of the skip rather than in it.

A screen was attached to the side bars above the skip, composed of light plates, inclosing three sides but open on the loading and dumping side. It reached down to the top of the skip, but was spaced far enough outside, so that the skip could turn through it to dump. Since there was a certain amount of danger in adding this complication to the skip, and the spilling was not greatly reduced, it was removed.

The spilled ore falls into a pocket at the bottom of the shaft, from which it is loaded into cars and hoisted on the cage to the 16th level bin. It amounts to about 17 lb. per skip, or 0.21 per cent. of the ore hoisted. It results partly from ore falling outside the skip when loading, partly from the ore which hangs up in the slide below the loading hopper and which falls after the skip is hoisted and partly from overloading. If the skip does not dump clean, and it is not noticed, there may be enough remaining to make an overload when the next charge is tripped into it. As a result of the spill, there has been heavy wear on the timber in the hoisting compartments.

The skips had originally only 70 cu. ft. capacity, and not sufficient margin to prevent an occasional overload. They were rebuilt and made to hold 90 cu. ft.

The skip is of standard Kimberly type, but built heavily. The dumping guides on the surface are formed of angles and heavy steel castings. As the castings wear, thin lining plates are riveted on.

Ore Loaded into Railroad Cars by Belt Conveyors and Trippers

The ore is dumped into a small bin holding about 12 tons, from which it is carried to the railroad track below, and loaded into cars for shipment to the reduction works. The arrangement of the conveyor plant is shown in Figs. 14 and 15.

A 36-in. steel pan conveyor forms the bottom of the bin, and in transporting the ore to an inclined belt equalizes the load. The receiving bin was made small to avoid a great accumulation of ore and consequent packing. It is now apparent that a larger bin would have been permissible and desirable.

The ore falls from the pan conveyor into a belt loader which breaks its fall and feeds it to the belt in the direction of belt travel (Figs. 16 and 17).



FIG. 14.—SACRAMENTO SHAFT LOADING PLANT IN 1909.

The V-shaped opening at the bottom of the pan allows the fine material to reach the belt first and form a bed for the coarser. The sides of the "V" are movable, and may be spread apart to widen the opening when sticky ore is hoisted.

Great difficulty was encountered in obtaining a feeder which would not clog with clayey ores. Even as it is, a solid mass of clay has accumulated above the plunger, the full width of the feeder, and several feet high, with the plunger still running ineffectively behind.

When the conveyor plant was first built, the ore fell from the pan conveyor upon the unprotected belt, which travelled on a slope of 15° . Boulders of ore sometimes bounded down the incline before they came to rest on a bed of finer material. When the feeder was put in, the upper 18 ft. of the belt was depressed to a horizontal position, and the bed of ore is well settled before it reaches the incline.

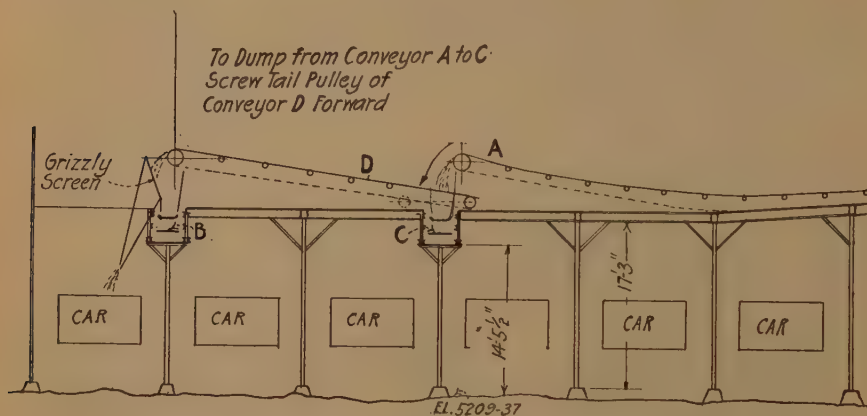


FIG. 15.—PORTION OF SACRAMENTO CONVEYOR SYSTEM.

The inclined belt A (Figs. 14 and 15) carries the ore down to the loading tracks, passing over a Blake-Dennison weighing machine on its way.

It is supported in a catenary curve to reduce the height of the trestle. This is now recognized as a mistake, as the maximum inclination is increased and the belt cannot be run taut without lifting it off the rollers when it is run empty, or with a light load.

The line of the incline belt crosses four railway tracks and delivers ore to conveyor C, the first of two conveyors parallel with them. The ore is discharged off the end of the belt and falls into a belt feeder similar to that at the shaft. When the ore is to be carried to belt B, the end pulley of the short belt D, the bearings of which are supported on slides, is screwed back by hand under the stream of ore discharged from A. It is then carried by the short conveyor to B and dumped into the third



FIG. 16.—BELT FEEDER.

belt feeder. The short belt thus takes the place of a tripper on a continuous belt and reduces the belt cost.

The loading belts *B* and *C* are each located between two tracks, and are long enough to load seven 50-ton cars and a sample car.

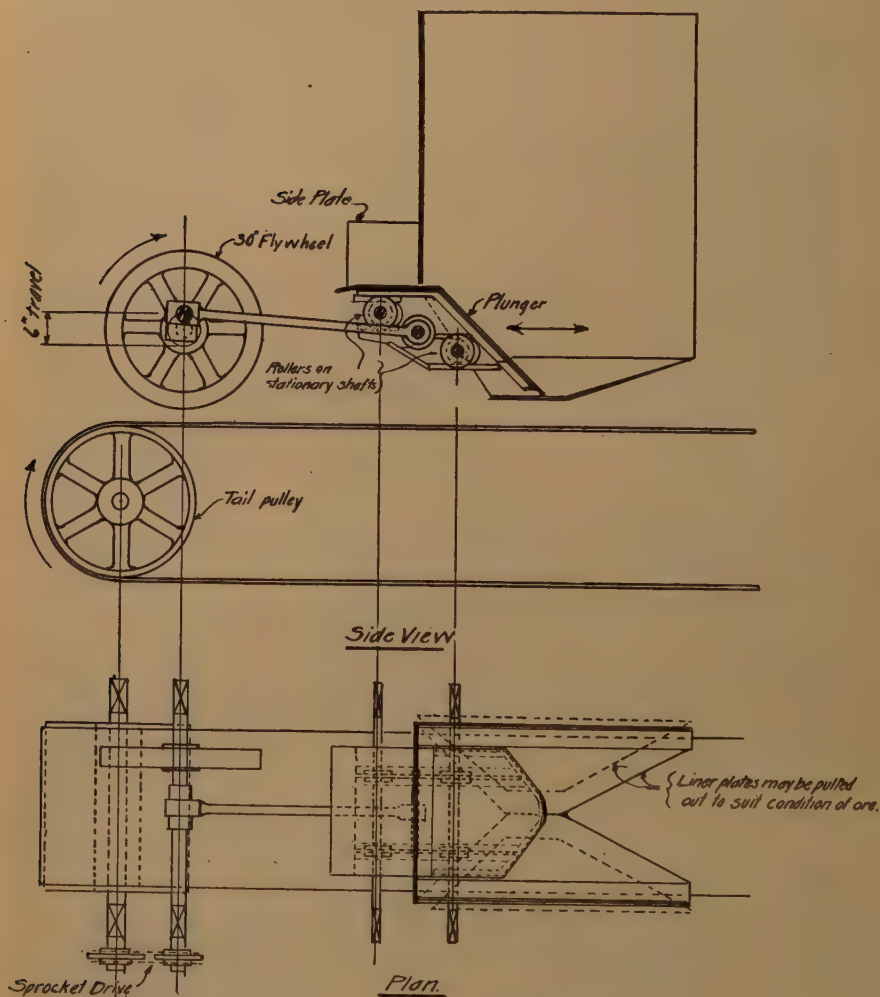


FIG. 17.—CONVEYOR PLUNGER FEED.

A tripper (Fig. 18) moves back and forth constantly, depositing layer after layer of ore in the cars below, until they are loaded to capacity as shown by the belt scales. At one end, it travels a few feet over the eighth car and this is taken as a sample of the lot. When the first lot is full, the sample car is switched to the other end of the loading shed and attached to a string of seven cars loading from the other conveyor,



FIG. 18.—BELT TRIPPER.

and the sample of another 350 tons is dropped into its opposite end. Its sample therefore represents two lots of ore of 350 tons each.

The belt speed may be varied from 240 to 360 ft. per minute, but rarely exceeds 300. The tripper speed is from 150 to 190 ft. per minute, but at the maximum speed of the belt would be 225. It travels back and forth from 36 to 40 times in filling a lot, depending on the weight of the ore; thus depositing from 72 to 80 layers of ore in the cars, and taking half as many samples while doing so. Although each lot may be of widely different composition, the ore in each car or in each linear foot of the car is the same as any other in the same lot. At the smelting plant, the ore is dumped into pits, one lot directly on the other, until a bed of 10,000 tons is made up. It has been found that when loading by steam shovel from one end of this bed, the ore encountered in working through to the other end is sufficiently uniform in composition to secure good metallurgical results. The sampling is also sufficiently accurate.

In its travel, the stream of ore from the tripper crosses a spout 12 in. wide, which catches a sample, splits it, returning one-half to the car, and drops the remainder into a heap on the ground, which is quartered down and assayed for quick returns. It is surprising how closely this extremely rough sample checks with the smelter returns. Although there is considerable variation in the sample of individual lots, the average for a month's run is rarely more than 0.1 per cent.

The first spout tripper was not positive in operation, and a man was needed to ride on it to keep it clear when loading mud. A special tripper was built, in which the ore is discharged upon a short 36-in. belt running at right angles to the main belt. It is driven by a small reversible motor carried on the tripper, which collects its power from a trolley wire, and may load into either of the string of cars spotted below. The spout tripper was built so high, to secure a steep incline for the spout, that it was unsteady and it was impossible to prevent the main belt from running out of line and cutting the edges. The belt tripper is low and steady and the main belt runs more truly and with less wear.

The tripper belts cost 0.266 c. per ton of ore handled. Now that there is a smaller proportion of muddy ore, a low, heavy, spout tripper might be better. Less attention than before would be needed to keep the spouts clear.

The inclined belt *A* is driven by a 30-hp. direct-current variable-speed motor at the lower end. The belt travel can be varied between 240 and 360 ft. per minute. The pan conveyor at the shaft is driven from the upper pulley by a sprocket chain, as is the belt feeder also. Belt *D* is also driven from this motor, through a friction clutch. Very little power is required, except to start, about $10\frac{1}{2}$ hp. with, and 8 hp. with-

out belt *D*. When loaded and in motion, it will drift so far that it is equipped with a brake in order to stop it quickly when necessary. The two loading belts *B* and *C* are driven by a 30-hp. variable-speed direct-current motor at the far end, through friction clutches. The tripper belts are driven by 5-hp. motors.

The trippers are pulled back and forth by $\frac{1}{2}$ -in. steel cables driven through reversing clutches, geared to the drive shafts of the belt. Buttons on the rope throw the clutches over at points of reversal, by turning the three-way valve of a compressed-air cylinder, the piston of which acts on the clutch levers. The buttons and points of reversal may be changed quickly, but there is little alteration in the routine and it is rarely necessary. The operation is positive and very satisfactory since powerful clutches have replaced the lighter ones formerly used. The conveyor belt passes idly through the tripper, and there should be less wear than if it supplied the power to propel it.

The control of the whole plant is located on a platform between the loading belts and near the incline. Here are the starting switches for all motors, brake for the incline belt, and levers connected by cables with the clutches at the driving end of the belt conveyors. The belt speed is governed by the speed of hoisting. They are run faster when hoisting from the upper levels. They are stopped when the skips are delayed or stopped for any purpose. The conveyor man is stationed at this point, where he can see all the belts and everything that goes on. One Mexican helper oils up rollers, chains, pulleys, etc. Another is stationed at the shaft to look after the loading end. A mechanic on day shift has been found desirable to attend to current repairs, to re-lace belts, etc. Other men are required from time to time to make more extensive repairs, clean tracks, attend to sampling, etc.

Many changes have been made in the original installation. Belt feeders have been added. The incline belt has been cut in two to eliminate a tripper. The trippers have been rebuilt. The control of the driving mechanism has been centralized. A roof has also been added to protect it from the weather.

The ore is rough, uncrushed, and very hard on belts. The life of belts is gradually increasing, due to improvements in the plant and a better understanding of the conditions.

Recently, it has been desired to screen a certain class of ore for fettling in the reverberatory furnaces, and to supply a coarse sulphide for pyritic smelting. This has been done by raising the discharge end of belt *D* so that the ore falls against a screen above the belt feeder. The fine ore falls into cars directly below, the reject into the belt feeder and makes up a regular lot and sample of coarse screened ore. Sufficient for present needs can be screened out in this manner, but if more were required, more costly changes would be needed.

Increase in Underground Storage Capacity Contemplated

After the changes in the shaft storage bins, no others were necessary to reach the capacity desired or increase the certainty of operation. Alterations have been for purposes of economy or convenience. There have been few delays caused by accident, and none which stopped operations for more than a few hours.

The most rapid hoisting in one day was 998 skips, containing 3,467 dry tons of ore, in three shifts of $7\frac{1}{2}$ hours net working time each. There was not enough ore to fill out the third shift at the same rate as the other two. The rate for an hour or two has exceeded this considerably. The maximum rate for any month was in February, 1914, when 65,901 dry tons were hoisted in 58 shifts, or 1,136 per shift. When hoisting at this rate, the shaft bins have not sufficient storage capacity. Time is wasted in changing frequently from level to level to avoid blocking the haulage motors. It is to prevent this that the 260-ton bin on the 400 was put in. Two others are in contemplation, which would increase the storage capacity to 1,350 tons. The hoisting capacity of the shaft would be increased, and there would be fewer delays to electric haulage.

Compressed-Air Hoists for Men and Timber at Most Shafts

The engine for men and timbers at the Sacramento shaft is a compound, two-cylinder engine, made by the Nordberg Manufacturing Co., but drives through Wuest herringbone gears. The cylinders are 18 in. and 30 in. by 42 in. stroke, the gears are 32 in. and 73 in. in diameter, with drums 9 ft. in diameter by 48-in. face, grooved for $1\frac{1}{4}$ -in. rope. The brakes, clutches, etc. are actuated by oil under pressure from the same accumulator which the ore hoist uses, and in a similar manner. The two triplex pumps to supply pressure for the auxiliaries have plungers $2\frac{13}{16}$ in. in diameter by 6 in. stroke, but the oil used when landing several decks is more than it can supply. Three-deck cages in balance are used.

When ore hoisting was transferred to the Sacramento shaft, the Holbrook, Spray, and Lowell steam hoists were connected up to drive by compressed air from the central power plant.

The Gardner hoist was at first driven by steam from the power house, delivered through a 5-in. main, 920 ft. long. Upon making tests on the efficiency of hoisting by compressed air, that was found to be more economical, and the Gardner hoist is now driven by air also.

No changes were made in the engines when using compressed air. The cylinders for brakes, clutches, etc. use cold air. The leakage has been less than when hot air was used and there has been less trouble with packing.

The air for the cylinders is heated to a temperature of 225° to 275° F. The cylinder of the heater is filled with heavy high-test oil, which flashes at about 700° F., and is inclosed in a brick setting like a small boiler. It is heated by a small coke fire under the front part, which is covered by a brick arch to protect the oil from overheating. The gases return to the stack in front through fire tubes in the lower third of the cylinder.

The air makes three passes through the cylinder in small tubes. An open pipe from the dome carries off any products of distillation. The temperature of the oil ranges from 250° F. to 275° F., although it may be carried higher without undue loss of its lighter elements.

The reheater supplies a reservoir of heat for the heavy drain on the system when the engine is starting, and does not over-heat if the engine is idle for a short time.

At the Lowell shaft, the air is heated to an average temperature of 250°. The air consumption there is estimated to be 30,000,000 cu. ft. of free air per month, at a gage pressure of 92 lb. In the past five months, the consumption of coke has been 3.8 tons per month.

The oil grows slowly thicker by the gradual loss of its volatile constituents. At a temperature of 275° F. it has been used for two years before getting too thick to circulate freely enough to keep the temperature of the whole quantity uniform, and avoid overheating a part of it. The usual life is 9 to 12 months. When too thick for further use, it makes an excellent rope compound. It preserves a rubbery consistency without hardening in cold weather, and does not melt. In fact, it has to be thinned somewhat before the rope takes it well.

Air Stored in Six Cylinders

An air-storage system, composed of six cylinders of 2,500 cu. ft. capacity each, is connected by a 24-in. pipe to a reservoir above. The reservoir holds 4 ft. of water, and is set at such a height that it exerts 93 to 91 lb. pressure on the tanks as it is full or empty. A drop in pressure from 93 lb. to 91 lb. will empty the reservoir and fill the air tanks with water, forcing 15,000 cu. ft. of air at 91 lb. or approximately 100,000 cu. ft. of free air back into the system. This eases the variable hoisting engine load.

Electric Hoist at Czar Shaft

The Czar engine was an old steam hoist for single-deck cages, built by the Union Iron Works, in San Francisco, about 30 years ago. The gear shaft was lifted out of its bearings, and replaced by another, geared to a 112-hp. alternating-current motor. It has since been replaced by an electric hoist driven by a 225-hp. alternating-current motor. It hoists a

double-deck cage at 800 ft. per minute, but it is usually operated in balance. It is driven through a direct line from the power house, where there is sufficient power in reserve to obviate the necessity of a fly-wheel set or other method of cutting down the peak load. It is started, however, by a General Electric automatic magnetic controller, to prevent too heavy a drag at the time of acceleration.

Brakes are operated by hand, the clutches by air pressure. Power for the latter was added after it was found that the method of throwing them by hand was too slow, when making many changes, as in cage hoisting it is necessary to do.

Several of the shafts are most inconveniently situated for waste disposal. At the Czar, Holbrook, and Lowell, the waste is dumped from the mine cars at the shaft into large cars holding 75 to 100 cu. ft., which are pulled out to the waste pile, by small hoists, and dumped automatically. One of them is pulled up an incline and returns by gravity, and one runs away by gravity and the empty car is pulled back by an engine.

Careful tests of the efficiency of different methods of hoisting were made, the conditions governing the tests being shown in the following paragraphs. Comparative results of the tests are shown in Table IV.

Conditions of Hoisting Tests

Test No. 1

Double-drum engine, reels 7 ft. diameter by 5 ft. face, $1\frac{1}{4}$ -in. rope. Driven by duplex, tandem compound, condensing steam engine, with Corliss valves.

Dimensions, 18 and 28 in. by 48 in. Cut-off by governor. Brakes, etc., actuated by hydraulic pressure, condenser and circulation pump by electricity.

Hoisting $3\frac{1}{2}$ -ton skips in balance.

Average depth of hoisting 825 ft.; maximum depth 1,580 ft.

Boilers isolated for test and feed water weighed.

Steam main 5 in. by 619 ft.

Condensation in main 625 lb. per hour.

Ore weighed on conveyor at shaft, on Blake-Dennison scales.

Work done in changing levels calculated from weight of skip.

Test No. 2

Double-reel flat-rope engine, ropes $\frac{1}{2}$ by $5\frac{1}{2}$ in.

Driven by a 20 by 60 in. simple, duplex, non-condensing engine, with Corliss valves. Cut-off by governor. Brakes, etc. actuated by steam. Boilers close to hoisting engine.

Hoisting triple-deck cages in balance from a maximum depth of 1,600 ft.

Boilers isolated for test and feed water measured.

Ore or waste weighed on scales at collar of shaft.

Work done in changing levels calculated from weight of cages, etc.

Steam for auxiliaries was generated in another boiler, and amounted to 16.1 lb. per shaft horsepower hour, making a total consumption of 111.6 lb.

Test No. 3

Double-reel flat-rope engine, ropes $\frac{1}{2}$ by $5\frac{1}{2}$ in.

Same engine as No. 2, and same hoisting conditions, but engine driven by compressed air.

Air measured by displacement of compressor run especially for this engine.

Allowance is made for volumetric efficiency. Transmission losses are practically all leakage, not included, since the proportion chargeable to hoisting depends on the quantity of air passing through the main.

Air heated by coke heater.

Auxiliaries for brakes, etc., driven by cold air.

Ore or waste weighed on scales at collar of shaft.

Test No. 4

Double-reel flat-rope engine, ropes $\frac{1}{2}$ by $5\frac{1}{2}$ in.

Driven by a 21 by 60 in. simple, duplex, compressed-air engine, with Corliss valves. Cut-off by governor.

Hoisting triple-deck cages in balance from a maximum depth of 1,032 ft.

Air measured by displacement of compressor run especially for this engine, allowance being made for volumetric efficiency and line leakage, which was measured.

Air heated by coke heater.

Auxiliaries for brakes, etc., driven by cold air.

Ore or waste weighed on scales at collar of shaft.

Test No. 5

Double-reel flat-rope engine, ropes $\frac{3}{8}$ by $5\frac{1}{2}$ in.

Engine driven by a 20 by 48 in. simple, duplex, compressed-air engine, with Corliss valves, with cut-off by governor.

Hoisting triple-deck cages in balance from a maximum depth of 943 ft.

Air measured by displacement of compressor run especially for this engine, allowance being made for volumetric efficiency and line leakage, which was measured.

Air heated by coke heater.

Auxiliaries for brakes, etc., driven by cold air.

Ore or waste weighed on scales at collar of shaft.

Test No. 6

Double-reel flat-rope engine, ropes $\frac{3}{8}$ by 4 in.

Engine driven by a 16 by 42 in. simple, duplex, compressed-air engine with Corliss valves; cut-off by governor.

Hoisting double-deck cages in balance from a maximum depth of 645 ft.

Air measured by displacement of compressor run especially for this engine, allowance being made for volumetric efficiency and line leakage, which was measured.

Air heated by coke heater.

Auxiliaries for brakes, etc., driven by cold air.

Ore or waste weighed on scales at collar of shaft.

Test No. 7

Double-drum geared engine driven by 225-hp. induction motor. Current three phase, 2,200 volts.

Started by automatic magnetic control.

Hoisting double-deck cages in balance from a maximum depth of 417 ft.

Power measured by wattmeters at power house, and includes line losses.

Every fifth car of ore or waste was weighed on scales at the collar of the shaft. Equivalent pounds of steam and cost of power at power house per shaft horsepower hour are high, due to inefficient turbines at power house and also low load factor.

Test No. 8

Conditions similar to Test No. 7.

TABLE IV.—*Tests on Hoisting Efficiency*

Test Number	1	2	3	4	5	6	7	8
Date of test.....	4-19-11	4-6-09	2-18-12	2-25-12	3-3-12	3-3-12	6-4-15	6-14-15
Duration of test.....	8 hr.	7½ hr.	2 hr. 4 min.	1 hr. 16 min.	49 min.	30 min.	6 hr. 30 min.	3 hr. 15 min.
Vacuum, inches.....	19.8							
Motive power.....	Steam	Steam	Air	Air	Air	Air	Electricity	Electricity
Pressure gage, pounds	137	125	89	89	87	88		
Temperature, ° F.....			205	150	220	200		
Work done, shaft horsepower hours.....	897.46	335.20	82.5	45.94	22.13	4.75	48.92	33.23
Power consumed per shaft horsepower hour, hoist alone.....	29.08 lb. steam	95.5 lb. steam	1,489 cu. ft. free air	1,966 cu. ft. free air	1,897 cu. ft. free air	2,322 cu. ft. free air		2.47 kw-hr.
Including condenser...	31.94 lb. steam							
Including line loss.....	34.29 lb. steam						2.93 kw-hr.	2.6 kw-hr.
Including auxiliaries...	34.29 lb. steam	111.6 lb. steam						
Including reheating...			1,602 cu. ft. free air	2,177 cu. ft. free air	2,037 cu. ft. free air	2,494 cu. ft. free air		
Equivalent pounds steam per shaft horsepower hour at power house.....	34.29	105.07	52.87	71.84	67.22	82.30	82.04	72.8
Cost of power per shaft horsepower hour....	\$0.0137	\$0.0399	\$0.0288	\$0.0392	\$0.0366	\$0.0448	\$0.0465	\$0.0416

In Test No. 2, the 111.6 lb. of saturated steam at 125 lb. pressure is reduced to 105.07 lb. in order to make an equitable comparison in efficiency of all tests. In all other tests, the equivalent pounds of steam are based on 150-lb. pressure with 75° F. of superheat, which are the power-house conditions.

In Tests Nos. 3, 4, 5, and 6, the cost of re-heating air appears as an amount of compressed air equal to it in cost.

In Test No. 7, the estimated line loss is 3 per cent.

It is assumed from power-house figures that 33 lb. of steam will compress 1,000 cu. ft. of free air to 93 lb. gage.

It is also assumed that 28 lb. of steam at the power house generates 1 kw-hr. at the power-house switchboard.

The cost of available steam at power-house is 45 c. per 1,000 lb.

The cost of air at the power-house is 1.8 c. per 1,000 cu. ft. of free air compressed to 93 lb. gage.

The cost of electricity at the power house is 1.59 c. per effective kilowatt-hour at the power-house switchboard.

The double- and triple-deck cage hoists are at a disadvantage compared to skip hoisting, since two or three lifts of 6 to 8 ft. must be made to land the decks. This only appears as so many feet of hoisting in work done, but it means getting the load in motion, which is the least efficient part of the work, twice or thrice instead of once.

This is exaggerated in the case of the Czar hoist by the shallow depth of hoisting. It will be noted that the decrease in power per shaft horsepower hour in test No. 8 as compared to No. 7 is due to a greater percentage of the material to be hoisted coming from the 400 level. When hoisting from that level, the current consumption per shaft horsepower hour was 2.29 kw.-hr.; from the 200 level it was 2.92 and from the 100 level 4.6 kw.-hr. per shaft horsepower hour. The record of this test is not representative of electric hoisting, and is included as a matter of interest only. Another hoist of the same type will shortly be erected at a 900-ft. shaft, and further tests will be made.

The simple steam hoist is inefficient at best, and particularly so in intermittent service, which does not, however, particularly affect either the compressed-air hoist or an electric hoist without fly-wheel motor generator.

It was a great economy to be able to use the hoisting engines as they stood, even if they were not quite so efficient as those especially designed for air.

TABLE V.—*Ore Haulage by Locomotive*

	1914, Cents per Ton	Four Months, 1915, Cents per Ton
Train labor.....	6.0	3.5
Loading labor.....	4.0	4.5
Motor maintenance.....	2.0	1.1
Car maintenance.....	2.2	1.3
Trolley maintenance.....	1.2	0.6
Track maintenance.....	2.0	1.4
Transfer chutes.....	2.2	2.8
Power.....	1.3	1.3
Miscellaneous.....	1.1	2.2
Foremen and bosses.....	1.0	0.9
	<hr/> 23.0	<hr/> 19.6

The average distance trammed per car in 1914 was 2,939 ft., returning empty.

Car repairs are rather higher than normal, since the equipment is growing old. Most of the cars have been in service for five years.

The charge for labor indicates that one man takes care of three to four miles of main trolley line, as well as bonding, and a proportion of the transmission line in the shaft. Trolley lines in heavy ground require much attention.

Track maintenance includes renewals and all other repairs. One man cares for $1\frac{1}{2}$ miles of track. Track and trolley extensions are not included. In 1914, this amounted to 5.3 c. per ton.

In 1914, 1.2 c. per ton was spent in transfer-chute construction, and an equal amount in 1915. Much permanent work has been done, and future costs are expected to be lower.

TABLE VI.—*Ore Trimming by Hand or Mule*

	1914	Four Months, 1915
	Cents per ton.	
Labor.....	10.8	9.5
Mules.....	1.0	1.3
Car repairs.....	1.6	1.3
Track repairs.....	3.2	3.3
Miscellaneous.....	3.3	2.8
Foremen and bosses.....	0.8	0.8
	20.7	19.0

The cost of hand and mule trimming is so intermingled that it is impossible to separate them. Ore is trimmed 194 ft. by hand and 708 ft. by mule. All new light track is charged to this account, whether for new drifts or replacements. In 1914, 47,800 ft. of drift were driven.

The cost of hoisting is divided into two classes, that of ore hoisting at the Sacramento shaft, and that of handling men, timbers, and waste rock at that and other shafts.

TABLE VII.—*Ore Hoisting, Sacramento Shaft*

	1914		Four Months, 1915	
	Total Cost	Cost per Ton	Total Cost	Cost per Ton
<i>Power:</i>				
Steam.....	\$12,632.12	\$0.019	\$4,216.43	\$0.023
Electric.....	436.72	0.001
<i>Attendance:</i>				
Engineers.....	5,694.54	0.009	1,848.25	0.010
Oilers, etc.....	3,017.44	0.005	931.92	0.005
Skip loaders.....	7,585.19	0.012	1,844.18	0.010
Cleaning stations.....	214.30
Miscellaneous.....	495.48	0.001	800.26	0.004
Hoist repairs.....	8,702.82	0.013	1,093.63	0.006
Cage repairs.....	2,549.03	0.004	625.60	0.003
Ore pockets.....	1,537.01	0.002	981.79	0.005
Hoist buildings.....	9.05	1.51
Ropes.....	3,252.43	0.005	586.39	0.003
Total.....	46,126.13	0.071	12,929.96	0.069
Tons hoisted.....	659,102	185,179

TABLE VIII.—*Conveyor-Plant Costs*

	1914	Cost per Ton Four Months, 1915
Power.....	\$0.0037	\$0.0027
Attendance.....	0.0122	0.0129
Belts.....	0.0026	0.0068
Other supplies.....	0.0007	0.0008
Repairs.....	0.0097	0.0078
	<hr/>	<hr/>
	\$0.0289	\$0.0310
Cost of screening.....	0.0015	0.0021
	<hr/>	<hr/>
	\$0.0274	\$0.0289

The rope cost includes that of some defective ropes on which a credit is expected.

Belts are usually charged off more rapidly than they wear out. A better idea of the cost is given by the records of the last belts on each conveyor.

TABLE IX.—*Conveyor-Belt Records*

Belt	A	B	C	D	Tripper
Width, inches.....	30	30	30	30	36
Length, feet.....	415	700	680	60	16 $\frac{2}{3}$
Tonnage handled.....	1,166,734	1,070,363	984,130	353,621	51,710
Cost per ton, cents.....	0.122	0.242	0.288	0.087	0.226
Cost per ton all belts, cents.....				0.965	

Cage riders were not customarily employed before the change in the hoisting arrangement. They were found to reduce the danger of shaft accidents.

It will be noted in Table X that the costs at the shafts handling no ore are much more in the aggregate than those at the Sacramento shaft. They are a necessary evil, to be held down to the lowest terms.

In 1907, the usual equipment for a producing shaft was: First-motion double-reel engine; double- or triple-deck cages; single-drum man engine; single-deck cage; boiler plant; blacksmith shop; drill- and tool-sharpening shop; timber-framing mill; drill repair shop.

In 1915, the plant is reduced to the first-motion hoist, to which an air heater has been added. The drill- and tool-sharpening shop, framing mill and drill repair shop were not eliminated by the change in hoisting methods, but followed the tendency toward centralization. The expense for surface labor has been reduced in proportion, although it appears principally as a credit to mechanical labor and other surface accounts. The

fuel consumption has been approximately cut in two, although the change rooms at each shaft must be heated by independent plants instead of by exhaust steam from the engines.

TABLE X.—*Cost of Handling Men, Timber, and Waste Rock*

	Uncle Sam	South-west	Czar	Holbrook	Spray	Gardner	Lowell	Sacramento Man Engine	Total	Per Ton
<i>Power:</i>										
Steam.....								\$7,634	\$7,634	\$0.011
Air.....	\$1,100			\$2,667	\$628	\$6,540	\$6,023		16,958	0.026
Air reheaters.....				565	158	911	492		2,125	0.003
Electric.....		\$744	\$1,472	71		747	234	243	3,512	0.004
<i>Labor:</i>										
Engineers.....	885	4,816	3,799	5,724	1,387	7,358	5,004	4,808	33,779	0.049
Oilers.....				1,662	548	1,782	1,997	1,666	7,655	0.011
Cagers.....	739	2,404	4,349	6,071	808	9,233	5,421	2,837	31,861	0.047
Cleaning stations.....			329	133	115	376	687		1,642	0.002
Miscellaneous.....				105					105	
Hoist repairs.....	886	1,036	2,416	1,475	519	2,283	2,550	1,695	12,860	0.020
Cage repairs.....	12	259	2,035	850	292	756	1,641	706	6,552	0.010
Rope repairs.....	90	93							183	
Ore pockets.....			116						116	
Hoist buildings.....			511						511	0.001
Rope maintenance.....			32	315	1	1,701	927	113	3,272	0.005
	\$3,712	\$9,352	\$5,059	\$19,638	\$4,456	\$31,687	\$24,976	\$19,702	\$128,582	\$0.189
Four Months, 1915										
<i>Power:</i>										
Steam.....								1,985	1,985	0.010
Air.....				858		1,927	1,626		4,411	0.022
Air reheaters.....				327		229	245		800	0.004
Electric.....		380	619	18		312	1		1,329	0.007
<i>Labor:</i>										
Engineers.....		766	1,216	1,848		2,209	1,266	1,172	8,478	0.042
Oilers.....				544		541	654	546	2,284	0.012
Cagers.....		640	1,600	1,621		2,036	1,216	992	8,106	0.040
Cleaning stations.....			176	116		43	87		422	0.002
Miscellaneous.....				89					89	
Hoist repairs.....		283	427	442		466	699	77	2,394	0.010
Cage repairs.....		53	345	192		855	1,283	457	3,185	0.015
Hoist buildings.....		23	10				27		60	
Rope maintenance.....		165	104	141		626	613	3	1,653	0.009
		\$2,310	\$4,497	\$6,196		\$9,244	\$7,717	\$5,232	\$35,196	\$0.173

As a whole, the costs of tramming and hoisting have not been reduced as much below the costs of 1906 as expected. A reduction in costs due to an improvement in method has been counteracted by the gradually increasing area of operation and necessary equipment. If a comparison

might be made with former methods, developed and extended to cover the present work, it would be all that was claimed for it and more.

Although a saving in repairs was counted upon in the original estimates, it was not specified. In 1914, underground repairs cost 51.2 c. per ton and in the first four months of 1915, 40 c. During the last three months of 1905, they cost 59.8 c. for labor alone, and for the first six months of 1906 the cost was 62 c. for labor and supplies.

Individual shafts are not so vitally essential as before. In 1911, the gases from a mine fire rendered the Lowell shaft impassable to men between the 800 level and the surface, for about five months. For two months, timber and supplies were lowered and waste was hoisted through the gas, but since no repairs could be made, a guide finally broke loose and jammed one of the cages. During this period, workmen were lowered through neighboring shafts and ore was hoisted as usual. Both production and costs for the year were normal.

The writer wishes to express his indebtedness to members of the mine department staff for assistance in collecting data used in this paper; W. Saben, Chief Clerk, Charles Legrand, Consulting Mechanical Engineer, R. E. Cameron, a member of his staff, and George Mieyr, Master Mechanic.

Ventilation of the Copper Queen Mine

BY CHARLES A. MITKE,* BISBEE, ARIZ.

(San Francisco Meeting, September, 1915)

INTRODUCTION

THE Copper Queen mine is composed of seven divisions which are operated through the following shafts:

Division No.	Shaft	Depth, Feet	Air Current
1	Uncle Sam	600	Downcast (shut down April, 1914, to curtail production)
	Southwest	600	Downcast
2	Czar	400	Downcast
3	Holbrook	600	Downcast
4	Spray	800	Downcast (shut down April, 1914, to curtail production)
5	Gardner	1,000	Downcast
6	Dallas	1,400	Downcast
	Lowell	1,600	Upcast
7	Sacramento	1,700	Downcast

The workings of the different shafts are connected by motor-haulage drifts on the even numbered levels. The general location of the orebodies and workings is illustrated by the vertical projection of orebodies of the district as shown in Fig. 1.

The Uncle Sam, Southwest, and Czar workings have many connections to the surface through raises and extensive cracks that were caused by moving ground. These divisions are cool and are ventilated entirely by natural means.

The Holbrook is ventilated partly by natural and partly by artificial ventilation, while the Spray, Gardner, Lowell, and Sacramento are ventilated entirely by mechanical means.

In some of the divisions mentioned above a considerable quantity of air exhausts through shafts of adjoining properties.

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CONDITIONS UNDER NATURAL VENTILATION

Several Mine Fires Occurred

During the last few years several mine fires menaced both life and property. The most serious mine fire was that of the Lowell, which occurred during 1911. The district where the fire originated was between the 1,000- and 1,200-ft. levels. The orebody contained a high percentage of sulphur, and the gob a large amount of pyrite; as stoping progressed the friction between the blocks of sulphides increased, due to moving ground. Oxidation of pyrite and timber was also an important factor. The temperature increased until the timbers took fire, which in turn started fire in the sulphide ore. The gases resulting from this fire, such as SO_2 , and distillation products of wood, came through the upcast shaft and caused considerable damage, thus putting the shaft out of commission. By reason of broken ground, the fire could not be sealed off at that time, which made necessary the repair of 800 ft. of the shaft by men using oxygen helmets.

Several fires also occurred at different times in some of the other divisions. They were particularly dangerous under conditions of natural ventilation as gases and smoke usually entered the workings. One of the reasons, therefore, in planning a system of mechanical ventilation was to provide a better means of control immediately after a fire started.



FIG. 1.—VERTICAL PROJECTION OF OREBODIES. Plane of projection bears S. $32^{\circ} 46' 50''$ E.

General Working Conditions Investigated

During 1912, the company decided to make an investigation of the general working conditions of the Gardner division. By reason of the scarcity of connections with other mines this division contained the poorest working conditions and presented the greatest problem for ventilation. Readings were taken on natural air currents as to their velocity, volume, direction, temperature, relative humidity, CO₂ content, etc. This indicated that even though the entire mine seemed to be hot the average temperature was not high, varying from 75° to 100° F. The relative humidity, however, was exceptionally high everywhere in the mine, ranging from 90 to 100 per cent.

The CO₂ was under 1 per cent. in the general run of working places and from 1 to 3.5 per cent. in exceptionally remote workings. The velocity and volume of the air currents were very low. Moisture was deposited on timbers and the sides of drifts, making the mine damp; a species of fungi covered a large percentage of the timbers. About 225 men worked in this division on each shift, and from the standpoint of efficiency the quantity of fresh air per man per minute was not all that was desired. Men in hot stopes had to be changed every two or three days on account of the heat, and in this way by the time a man was familiar with the workings of his stope he was put into another place. Consequently, all the men who had fairly good working conditions had to share the ordeal from time to time, by working in these undesirable places.

It was the custom when conditions became particularly bad to drive new air connections. In a few instances they proved beneficial, but in the large majority of cases the quantity of air which passed one way or the other was so small that the effect was negligible.

THE RELATIVE MERITS OF PRESSURE AND EXHAUST SYSTEMS CONSIDERED

At first, plans were formulated and data obtained, as described under "Planning the System of Mechanical Ventilation," for both the pressure and exhaust systems. After the merits of each were carefully considered, the pressure system of ventilation was adopted. According to this system, all mine fans were placed underground on the deepest working levels of the different divisions so as to draw the air down the intake shafts, force it through the workings and stopes, and exhaust it to the surface through upcast shafts.

Four Reasons for Adopting Pressure System

The reasons for adopting the pressure system instead of the exhaust are as follows:

1. *More Practical.*—Since all the deep shafts of the Copper Queen are in operation and a large amount of work is done around each, as for example, men going on and coming off shift, handling timber, tools, and supplies, etc., it would not be practical to have an exhaust fan on the surface near the collar of the shaft.

In the Gardner, a subway below the collar of the shaft would be out of the question. Consequently, a large air-tight chamber, inclosing nearly the entire head frame, would have to be built if a suction fan were used. This arrangement would be expensive and unsatisfactory.

2. *Better Control of Mine Fires.*—The pressure system is the more advantageous in case of mine fires. With this arrangement, the entire workings are under pressure and the gas or smoke from a mine fire may be coursed to exit through neighboring shafts that are not in operation. Since the air is under considerable pressure at the bottom and sides of the present fire districts and each of the outlets over them is provided with regulator doors, the quantity of air passing through the fire zone can be increased or diminished at will and the fire consequently can be brought under control more readily than with the exhaust system.

3. *Greater Efficiency.*—Since the Spray, Gardner, Dallas, and Sacramento shafts are natural downcasts, the mine fans were placed underground so as to take advantage of these circumstances and multiply the volume of the natural downcast from five to ten times. The pressure system, therefore, has the advantage of the additional natural ventilation, which evidently would be lost with the other system; and more power would be required for the same quantity of air if the natural downcasts were changed to upcasts by suction fans.

4. *Greater Safety.*—Experience in the Lowell mine fire showed that in the event of the connections being closed between the fire and the suction fan, there is a possibility of smoke and gases entering the workings below and endangering the men. In the adopted system, the maximum air pressure is below the fire district, which makes it impossible for gases to go down, even if bulkheads of a sealed fire district should be opened suddenly by caving ground.

At present a churn-drill hole is being put down to a depth of 1,100 ft. over the old Lowell fire stopes. When this is completed water will be run in to cool the district. No trouble is anticipated since this was thoroughly tried at the Holbrook, where a large amount of water was turned into churn-drill holes and no smoke or gases were taken down against the pressure of the mine fan.

EQUIPMENT AND GENERAL ARRANGEMENT FOR MECHANICAL VENTILATION

The following pages give an outline of the general mechanical ventilating system which was adopted at the Copper Queen mine.

Combined Natural and Mechanical System in Holbrook and Gardner Divisions

Because of the many openings to the surface, the natural ventilation is satisfactory from the surface to the 400-ft. level in the Holbrook division. Below this level mechanical ventilation is used, as indicated in Fig. 2. There is a 20-in. blower on the 600-ft. level, which delivers 8,000 cu. ft. of air per minute. Since very little stoping is done between the 400- and

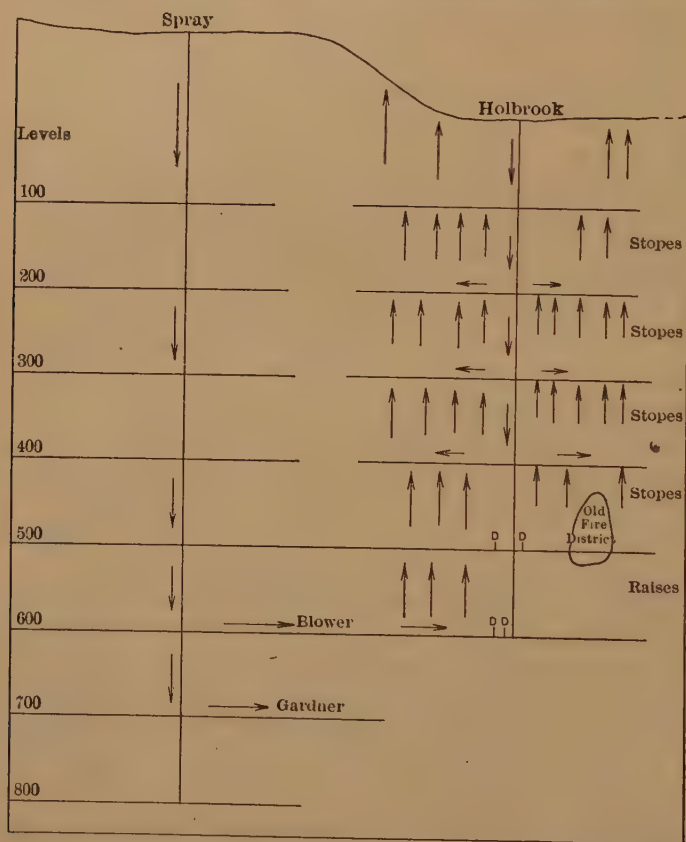


FIG. 2.—NATURAL AND MECHANICAL VENTILATION OF HOLBROOK DIVISION. VERTICAL CROSS-SECTION.

600-ft. levels, the blower is operated principally to keep a pressure around the old fire district to prevent any gases which may escape from entering the workings.

The outlines of the natural and mechanical ventilating system of the Gardner division are shown in Figs. 3 and 4. The arrows indicate the direction of some air courses and *DD* represents a few double doors. On the 900-ft. level, there is a Sturtevant mine fan delivering 72,000 cu.

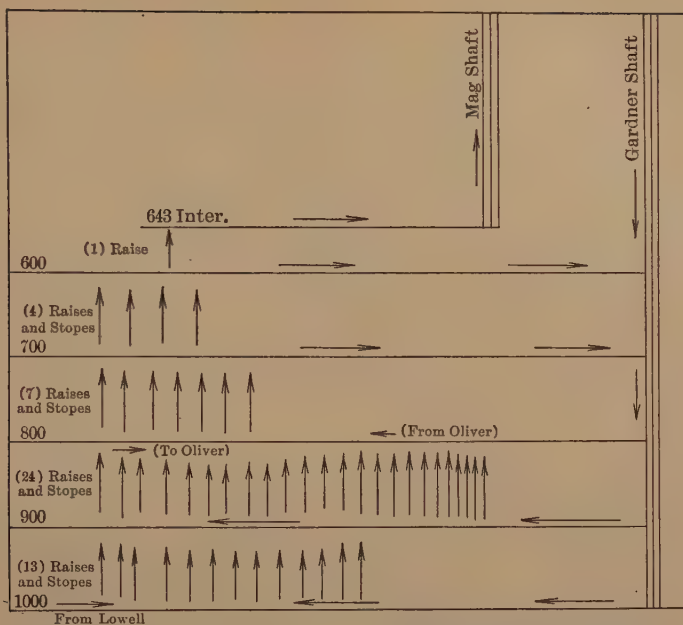


FIG. 3.—NATURAL VENTILATION SYSTEM OF GARDNER MINE.

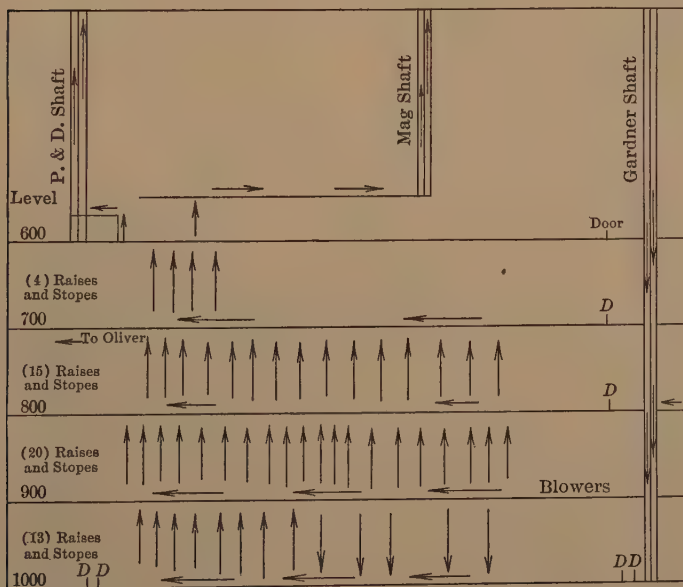


FIG. 4.—MECHANICAL VENTILATION SYSTEM OF GARDNER MINE.
VERTICAL CROSS-SECTION.

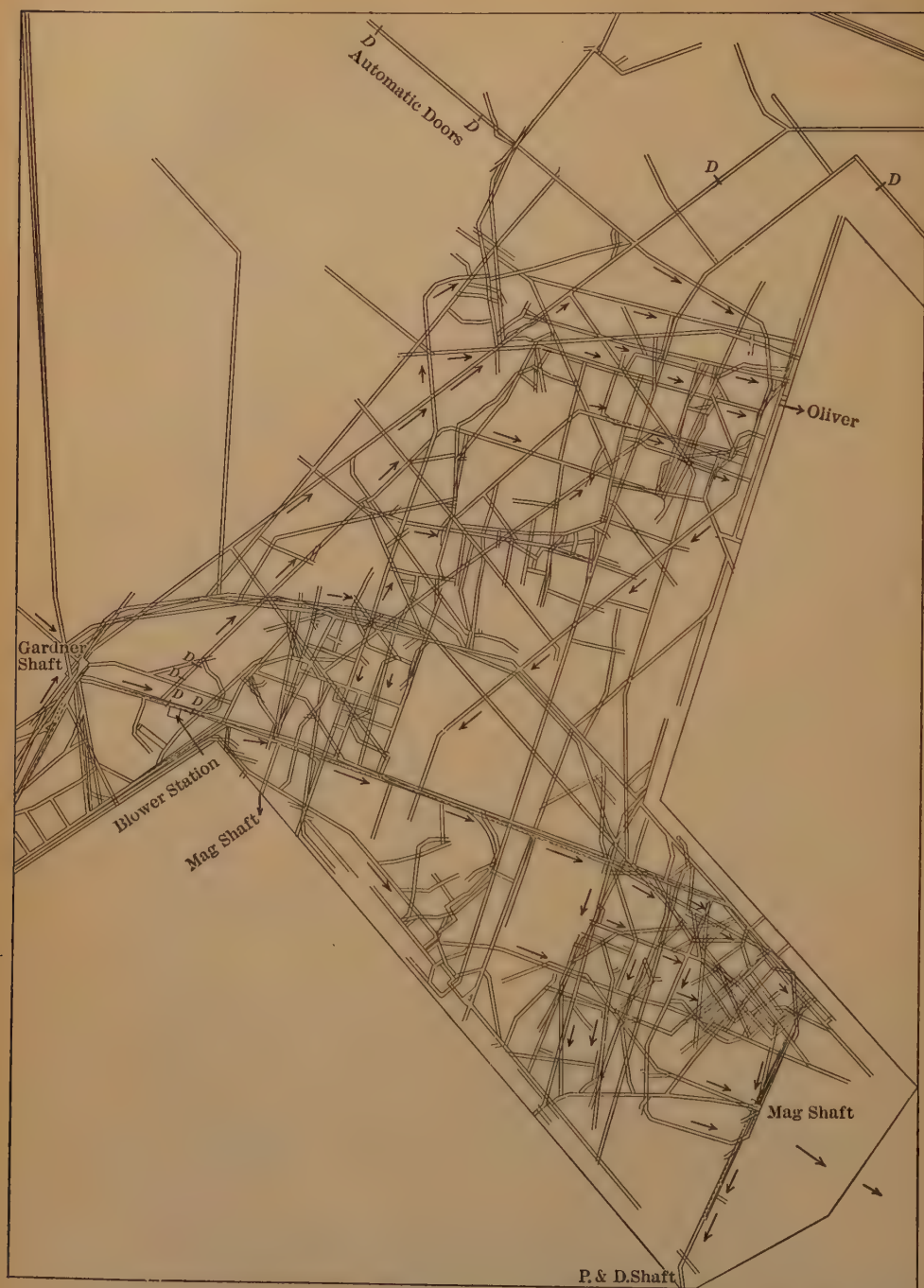


FIG. 5.—ARROWS INDICATE DIRECTION OF SOME IMPORTANT AIR CURRENTS.
HORIZONTAL PROJECTION OF LEVELS—GARDNER DIVISION.

ft. of air per minute against $1\frac{3}{8}$ in. of water pressure. About 32 hp. is being used. The combined level map (Fig. 5) illustrates the new connections which had to be driven, also the various directions of air courses when superimposed on one plane. There are 22 hand doors and two large doors on motor tracks, called automatic doors.

The mine fan draws the air down the Gardner and Spray shafts and forces the total quantity into the workings on the 900 level. Part of this air goes down to the 1,000 ft. and later comes up to the 900 ft., from whence all the air passes through stopes to the 800-, 700- and 600-ft., levels, and exhausts through abandoned workings and upcast shafts. On the 1,000-ft. level the pressure is built up by the use of automatic doors. These doors not only hold the pressure, but also allow the air to circulate freely on the 1,000-ft. level. The usual speed of motor trains passing

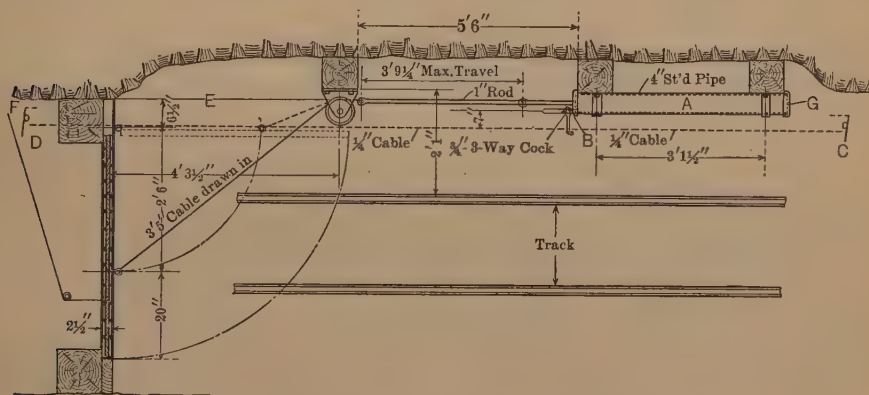


FIG. 6.—PLAN SHOWING AUTOMATIC DOOR ARRANGEMENT.

through these doors is about 7 miles an hour. The doors are 350 ft. apart. The arrangement for automatically opening the door is shown in Fig. 6.

Operation of the Automatic Door.—When the motor comes in the direction of C to D, the motorman reaches out and moves the lever C in the direction the motor is going. This opens the 3-way valve B, admits compressed air into the cylinder A and opens the door. When the train has passed through the door the motorman throws the lever D, which releases the compressed air and the door closes, assisted by the weights E and F. The valve G controls the speed of opening and closing of the door. The wires between C and D are crossed so that the levers are always moved in the direction the motor is going in opening and closing the door. The distance between lever C and the door is 70 ft., which allows ample time for the door to open when the motor is coming at an ordinary rate. The distance between D and the door is 175 ft. This makes it possible for a trip of cars to be in front of the motor on the re-

turn and at the same time allow the door to be opened. The cost of upkeep is small and the entire equipment only requires the amount of attention usually given to any machine.

The automatic doors are used on motor tracks only, while small hand doors are used in all ordinary drifts. The standard hand door is 6 ft. 6 in. high, 3 ft. 9 in. wide and $2\frac{1}{4}$ in. thick, made up of 1-in. boards with paper between the two thicknesses. The posts of the door frame are 6 by 10 in. by 6 ft. 6 in. and the cap 6 by 10 in. by 4 ft. 3 in. The motor

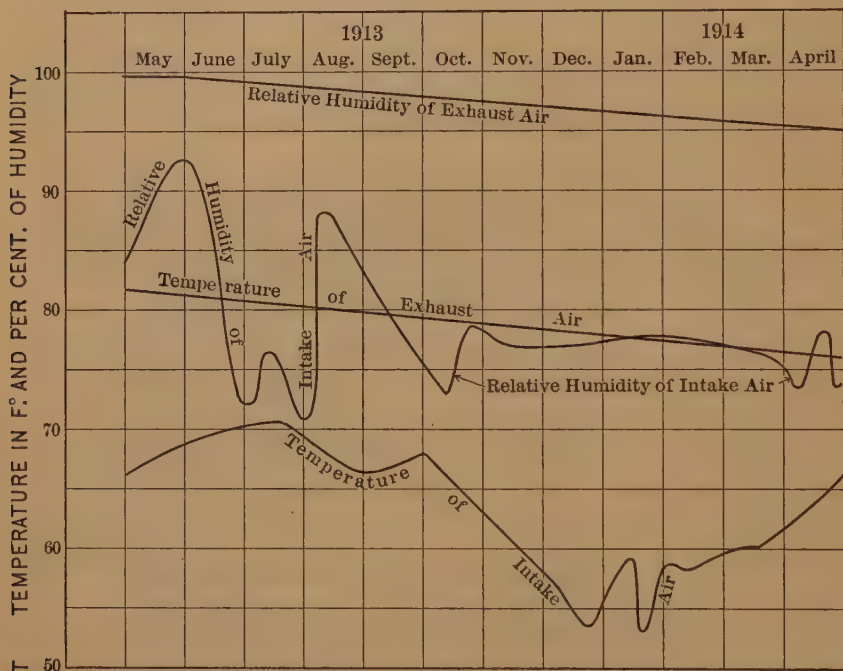


FIG. 7.—CHART OF TEMPERATURE AND RELATIVE HUMIDITY OF INTAKE AND EXHAUST AIR BEFORE AND AFTER NEW VENTILATION SYSTEM WAS IN OPERATION.

tracks are mounted on a timber sill set in concrete. The space between the bottom of the door and the sill is filled with concrete, except for clearance spaces for the flanges of the car wheels. The spaces between the door frame and the walls and roof are likewise filled with concrete. The automatic door is 6 ft. 11 in. high by 4 ft. 4 in. wide made of 1 by 12 in. boards faced on both sides with $\frac{3}{4}$ by 2 in. flooring.

Air Jets Used in Hot Workings.—Formerly small electric blowers were used to ventilate hot raises and stopes. These poorly ventilated places were usually some distance from the shaft, which required a considerable investment and long electric cables. Instead of using these blowers an air jet was designed and tried for workings of this type. The jet had

eight $\frac{1}{16}$ -in. holes and was placed in the center of a 14-in. funnel which was connected to a 10-in. pipe. About 800 cu. ft. per minute was delivered at the end of 50 ft. of pipe of this kind. The cost of compressed air is comparatively small and is less than one-third the cost of electrical power, when used in these particular instances. Of course the jet is only used until the connection is made. After connections are made considerable quantities of air pass through these drifts and raises from the large blower, and the ventilation is generally satisfactory.

New System Lowered Temperature and Relative Humidity

The chart, Fig. 7, illustrates the decrease in temperature and relative humidity of the intake and return air before and after the blowers were started. On account of the large amount of moisture in the ground and also the wet conditions of moist drifts and workings, it took about four months to dry the timbers and working places after the new ventilating system was in operation.

Velocity and Volume of Air Increased

The blowers were started June 1, 1913, resulting in an increase in velocity of the intake air from 65 ft. per minute on June 1, to 1,000 on June 30 and 1,160 on July 31. The volume also increased from 10,000 cu. ft. per minute on June 1, to over 60,000 on June 30 and 70,000 on July 31. The reduction in the temperature and humidity was not so marked as the increase in velocity and volume. The division in general, however, seems much cooler and the environment is comfortable.

Lowell and Sacramento Divisions Ventilated Jointly

The natural and mechanical ventilation systems of these divisions are outlined in Figs. 8 and 9. There is a Sirocco blower (capacity 35,000 cu. ft.) on the 1,400-ft. level of the Dallas shaft, forcing the air towards the Lowell against a $1\frac{3}{8}$ -in. water pressure and another blower on the 1,600-ft. Sacramento (capacity 45,000 cu. ft.) operating against a $1\frac{1}{2}$ -in. water gage. The reason for the higher pressure in the Sacramento is to have the air from that level join the Dallas air on the 1,400-ft. level and from there rise through the workings to the 900-ft. level of the Lowell, and exhaust through the two shafts as indicated in Fig. 9.

Part of Fire District being Cooled.—An example of the effect of large volumes of air passing through a district, part of which is on fire, may be obtained from the 13-10-10 stope. Here the temperature has been reduced from 87° to 83° F. To accomplish this result, 15,000 cu. ft. of air per minute was driven through this stope from Sept. 1, 1914, to Feb.

1, 1915, a period of five months. Before this blower was started the humidity was about 98 per cent. and the velocity of the air was scarcely noticeable. After the 15,000 cu. ft. of air per minute was driven through the stope, even though the temperature showed a very little difference, the humidity dropped to about 80 per cent. and the velocity increased to about 400 ft. per minute. As a result the stope seems cool and the men can do efficient mining. This small reduction in temperature also shows that it requires a large quantity of air circulating a long time in a fire district to cool the country rock.

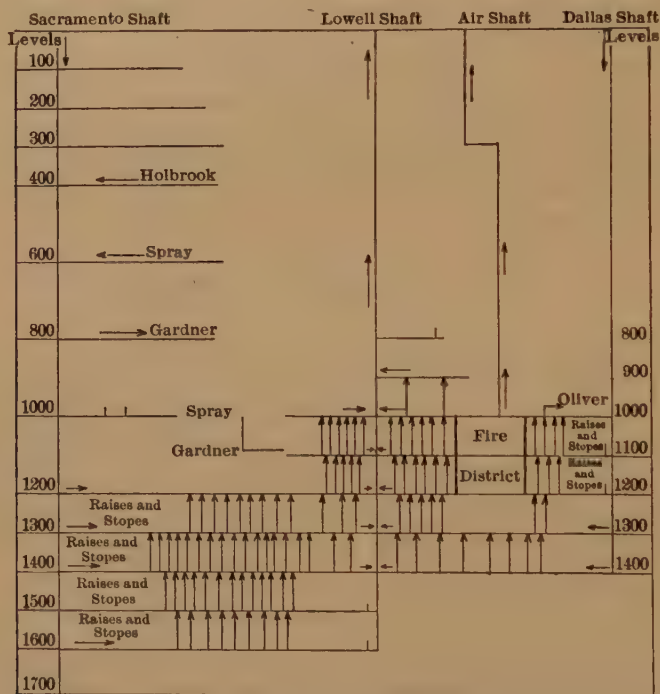


FIG. 8.—OUTLINE OF NATURAL VENTILATION SYSTEM OF LOWELL AND SACRAMENTO MINES. VERTICAL CROSS-SECTION.

The velocity and volume of air have been increased through all the stopes and workings and although the temperature throughout the entire mine has not been lowered much, nevertheless the workings seem much cooler on account of the increased velocity and volume and decrease in humidity.

DATA COLLECTED BEFORE MECHANICAL VENTILATION SYSTEM WAS PLANNED

In order to plan the general system of mechanical ventilation of the Copper Queen, as outlined above, the following methods of obtaining

standard was used in all drifts, raises, and places where there are air currents that can be detected and measured with the anemometer. This method is also the more important because the increase of velocity of air materially affects the efficiency of the men.

The quality standard is very expensive and its application is necessary only in exceptional cases. It has only a remote bearing on the efficiency of the men, but occasionally it is found advisable to have accurate determinations made on several samples of mine air.

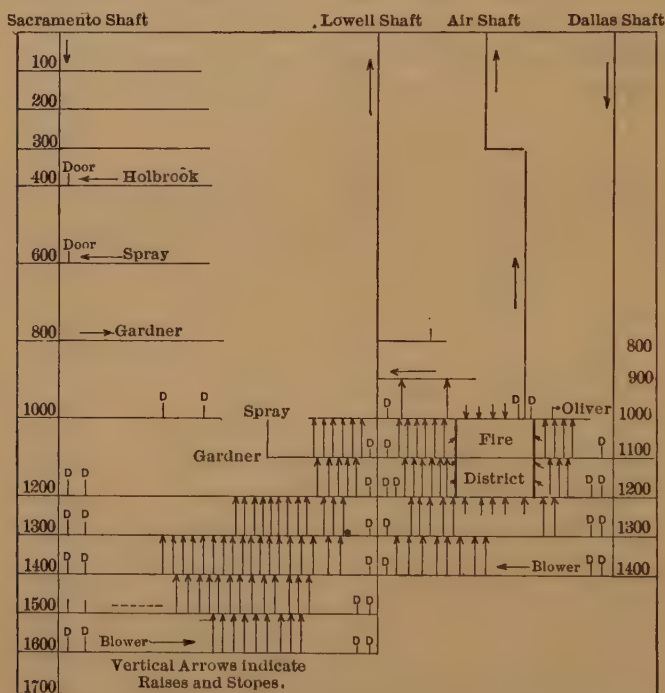


FIG. 9.—OUTLINE OF MECHANICAL VENTILATION SYSTEM OF LOWELL AND SACRAMENTO MINES.

Averages Plotted on the Mine Map

The data secured as explained in the paragraph on "Recording of Data" were averaged every week. Blueprints of the different levels of the mine were made and the monthly averages written on them in their proper places. The direction of air currents was also indicated by arrows, especially where readings were made with the anemometer.

Amount of Air Necessary for Each District Calculated

In order to calculate the amount of air necessary for each district, it was necessary to take into account the following:

1. Number of men and animals in each district.
2. The production of CO₂, or other gases, as shown by the chemical analyses.
3. Relative humidity.
4. Temperature.
5. Amount of explosives used.
6. The distance from currents of good air.
7. The number of lights.
8. Air leakage.
9. Friction of the air currents.
10. Number of splits of the air current.
11. Method of distribution.

Natural Ventilation Utilized Where Possible

In order to get the most economical system, it was necessary to utilize all possible natural ventilation and secure the proper distribution of the air. The latter is extremely important because the efficiency of the ventilating system depends largely on the proper distribution of the air currents. The arrangement of air currents by mechanical means was designed to coincide with that of natural ventilation in nearly all cases.

Natural Ventilation Supplemented by Mechanical Means

In the divisions where natural ventilation was not sufficient, the natural air currents were strengthened by mechanical, or artificial means, by installing large mine fans.

A Good Working Atmosphere Dependent on Several Factors

There is a diversity of opinion among authorities on what constitutes "a good working atmosphere," and to what extent it will be economical to ventilate a mine, considering the cost of installation, maintenance, and power. The quantity of air necessary for the ventilation of one mine would be inadequate for the ventilation of another. The same can be said regarding the quantity of air per man per minute, consequently every mine is an individual problem, and while generalities are made, to get the greatest efficiency of a ventilating system, a detailed study of the mine must first be made, and the new ventilating system adapted to meet the particular needs of each mine.

In a number of States the quantity of air in coal mines required by law is usually specified, as, for example: The Anthracite Mine Law of Pennsylvania specifies a minimum quantity of 200 cu. ft. per man, per minute. One of the modifying clauses of the above law stipulates that the amount of air in circulation should be sufficient "to dilute, render harmless, and sweep away, smoke and noxious or dangerous gases."

Even in coal mines, therefore, where the problems of ventilation have not the intricacies found in metal mines, there is no fixed standard upon which to base figures for "a good working atmosphere" or an efficient ventilating system.

Experience in ventilation of the Copper Queen mine has shown thus far that in poorly ventilated areas, comfort, efficiency, and economy are obtained by increasing the natural ventilation with mechanical means until the following conditions are obtained: Temperature, 77° F.; relative humidity, under 85 per cent.; velocity of air in working places at least 100 ft. per min.; volume of air in working places at least 250 cu. ft. per man, per minute. This may be termed a good working atmosphere and is the present standard for this camp.

In general, if the above velocity and volume are maintained in any working place it will not be necessary to make chemical analyses, because the percentages of CO₂, CO, NO₂, and SO₂ gases, etc., as well as the temperature and humidity, will be low and require no attention, except in a few instances.

Increased Velocity of Air Improves Working Conditions

Formerly there were several examples of stopes in the Copper Queen mine which were similar to the following:

In No. 8-13-5 stope of the Gardner division, the readings in working places for May 28, 1913, were

Velocity.....	10 ft. per minute (approx.)
Volume.....	10 cu. ft. per min. (approx.)
Temperature.....	84° F.
Humidity.....	96 per cent.

A small 5-hp. blower was forcing air into the stope. The miners were uncomfortable in this stope and complained about the excessive heat.

On Oct. 1, 1913, after the new ventilating system was installed, the following readings were made in the same stope:

Velocity.....	100 ft. per minute
Volume.....	500 cu. ft. per min.
Temperature.....	84° F.
Humidity.....	90 per cent.

The same miners mentioned above happened to be working in this stope on Oct. 1, 1913, and they remarked "the stope is not near as hot as it used to be and we can work much better." The men did not sweat excessively in this atmosphere, and the greater comfort experienced by them was due to the *increase in velocity*, which also means an increase in volume, with lower humidity, and not to the temperature, as it was the same in both cases.

When miners work in hot mines, drifts, and stopes, they generally neglect their personal safety. It was a frequent occurrence for the shift boss to enter a working place and after asking miners to bar down the back to discover that they had been working under loose boulders for several hours. It is evident, therefore, and proved by the rate of accidents, that men are on the alert and look out for personal safety far more in a comfortable atmosphere than in hot, damp, and disagreeable places.

In regard to efficiency, the tonnage per man shift has increased with the increase in the velocity and volume of air resulting from the installation of large mine fans. The greater efficiency of the men may be partly assigned to better organization and cooperation and partly to the psychological effect resulting from good ventilation. For instance, in the language of the average miner, he "tries to make a showing every day." In a comfortable atmosphere, he will take an interest in his work, and go about it in a more contented spirit than he could possibly do under hot, unfavorable, unhealthy, and depressing conditions.

A special effort, therefore, is constantly being made at the Copper Queen to increase the velocity of the air in all the raises, drifts, and stopes, and keep the ventilating system up-to-date with the newly developed workings.

CHANGES AFTER MECHANICAL VENTILATING SYSTEM WAS COMPLETED

Development Work and Ventilation System Planned Together

In general, the mine department makes an effort to plan development work in line with the ventilating system, as well as for prospecting purposes. The drifts are usually driven about 5 by 8 ft. and raises are run two compartments each, 50 in. in the clear. Recently standard raises have been adopted throughout the mine, and the timber compartments as well as manways have also been standardized. Rails are put over timber compartments to allow free passage of air (see Fig. 10) and manways have been made larger to permit more air to pass through them and at the same time to make it easier for the men to go up and down. Also when an orebody is found it is properly developed and prospected before stoping is commenced. In this way ventilation is really first established, while the orebody is being brought into a condition for stoping operations. This is a great help in the ventilation of stopes.

Square-Set and Shrinkage Stopes Ventilated Easiest

Recently other methods of stoping besides the original square set have been giving good results. Shrinkage, cut-and-fill, and top-slice stopes have progressed far enough to show that each method has its

particular application to certain kinds of ground. In ventilating stopes under these systems, probably the most easily ventilated are the square-set and shrinkage stopes. Next to these are the cut-and-fill, and the most difficult the top-slice stopes. However, in all cases at present where the top slice is being used, separate drifts and raises were driven to facilitate

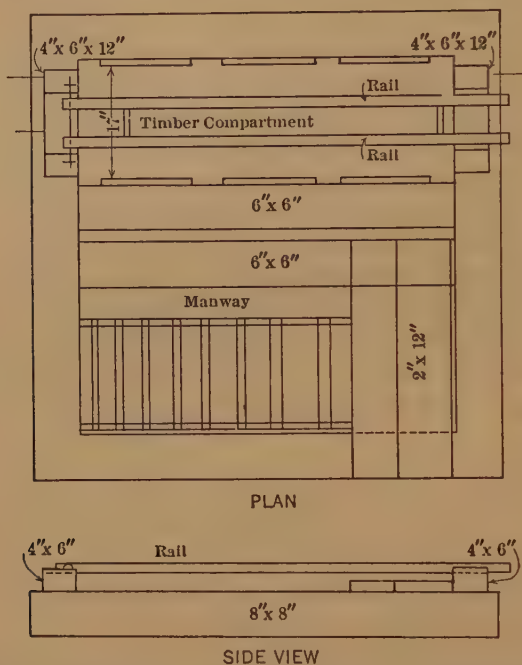


FIG. 10.—METHOD OF COVERING TIMBER COMPARTMENTS.

the handling of timber and tools and at the same time to give connections for ventilating purposes. Consequently, the top-slice stopes have about the same temperature and humidity as found in the square-set sections.

ADVANTAGES OF NEW VENTILATING SYSTEM

1. A smaller quantity of compressed air is used for ventilation.
2. All possible natural ventilation is utilized and the general direction of air currents is the same for both natural and mechanical ventilation.
3. There is a decrease in temperature and a lower percentage of humidity and CO_2 throughout the mine, which means healthier working conditions.
4. A saving of eight 5-hp. blowers was made, as these have been removed from the mine.
5. The five lower divisions are under pressure, which partly acts as

a check in preventing some mine fires from crossing the Copper Queen side lines, as evidenced in the proximity of one mine fire near the Gardner workings.

6. Less decay of mine timbers.

7. The volume of air has been increased from 60 cu. ft. of air per man per minute, to about 300 cu. ft.

8. Formerly a large amount of moisture kept the timbers and working places in a damp condition and a species of fungi also covered most of the timbers. Since the mine fans were started, a large amount of water is removed daily from each division by the air currents so that the mine is comparatively dry and the fungi have disappeared.

9. The increase in velocity and volume of air resulted in the greater efficiency of the men.

10. The new ventilating system is advantageous to the mines adjacent to the Copper Queen mine; there is a great deal of broken and stoped ground along side lines permitting a large amount of air to escape through these avenues and ventilate stopes beyond the Copper Queen boundary.

DISCUSSION

GERALD SHERMAN, Bisbee, Ariz.—Mr. Mitke justly might have made much stronger his claim for increase in efficiency of labor due to the introduction of forced ventilation. The product per man shift in stoping in the Gardner division has practically doubled. While it is true that improved methods have been responsible for a great part of this gain, and what is due to better ventilation cannot be definitely known, yet it is also true that the results could not have been obtained in that division as it was in 1912.

The paper discusses the two systems of ventilation in use, exhaust and pressure. In both the object is the same, to take foul air or gas out of the mine by the most direct course and replace it by fresh. In coal mines, the exhaust system is in almost universal use, and since the most advanced work in ventilation has been in that class of mines, it was naturally the first one to be considered. It has the advantage of leaving all adits, shafts, or other openings free for use. Doors are only needed to direct the course of the air, and not to confine it.

Its disadvantages appear in case of fire, or if the ground over the mine is much broken by subsidence. The fan must work in gas in the case of a fire, and may be damaged or destroyed either by heat or by corrosive gas. At the time of the fire in the Lowell division of the Copper Queen, a small exhaust fan was placed over a ventilating raise, but the vanes and housing were destroyed by sulphurous gases within a few weeks.

If the ground above the mined area is much broken, as is frequently the case when the ore deposit is thick vertically, a good deal of air is

likely to pass through it, particularly if it is near the surface. Leakage from this source is a loss to the exhaust system by short circuiting, which is exaggerated by a contraction or block in the air course. It serves as an added outlet in the pressure system. This is strikingly shown at the United Verde, where many jets of steam may be seen coming up through the ground above the stoped country on a cold day.

If there is a fire, a cave in the air course between it and the exhaust fan would be a serious matter, while in the pressure system, the gases might be forced out through the broken ground, and the pressure could be raised if necessary to do so.

Mr. Mitke briefly mentions that repairs were made in the Lowell shaft by men using oxygen helmets. I believe that members of the Institute would be interested to hear a little more of this work.

In the Lowell fire of 1911, the caving of an air course diverted sulphurous gas to the shaft at the 800 level, through which it passed to the surface. As all mining was below this level, workmen entered the mine from other shafts, but for some little time cages were run through the gas, carrying waste rock, timbers and other supplies. The lag screws, pipe and cable hangers wasted rapidly by corrosion, the guides loosened and both cages finally jammed, although they were afterward pulled out. Mr. Mitke was borrowed from the Stag Canon Fuel Co. at Dawson, N. M., one of the Phelps, Dodge properties. He trained a crew of miners in the use of oxygen helmets and, by working from the cages, secured and replaced guides, etc., cut out dead power cables and generally repaired the shaft through the gas. It took three weeks of work, in two shifts, with frequent relays of men. That it was done successfully, and without accident, is due to Mr. Mitke.

JOSEPH P. HODGSON, Bisbee, Ariz.—Since artificial ventilation has been installed at the Copper Queen mine, the benefits have been so great for both the men and the property that, while other things besides ventilation and changes in mining methods have entered into the problem, we are anxious and willing to have a large share of the credit bestowed on the ventilation system. It was adopted after careful thought and much study, and we are well pleased with it. If we were to extend our ventilation system, or to put one in anywhere else, we would certainly install the system we have adopted, as it gives a large amount of air per man per minute, and we have the mine at all times under pressure so that we can control it in case of fire.

Mine Pumping

BY CHARLES LEGRAND,* DOUGLAS, ARIZ.

(San Francisco Meeting, September, 1915)

THE problem of mine pumping is so much affected by local conditions, and those conditions are so liable to changes during the life of a mine, that the best system to use is difficult to determine. The experience of the writer has been that, in general, for copper mines, electric pumping is most satisfactory unless the quantity of water to be pumped is great or the mine does not use electric power for any other purpose. As a rule, however, figuring on the cost of necessary power plant, the total cost of installation is greater with electric pumps than steam pumps.

As the water is liable to be gritty, outside-packed plunger pumps are to be recommended, and if the water is not acid, or only slightly so, chilled cast-iron plungers pay for their extra cost very quickly in diminished cost of packing. With chilled plungers it is possible to use metallic packing instead of soft packing, if the water is not too gritty or the lift too high.

In vertical pumps this packing is satisfactory for lifts up to 400 ft., in the experience of the writer, and may be found satisfactory for higher lifts.

For water slightly acid, cement-lined pump bodies have given good satisfaction.

For acid water, both plungers and pump bodies should be made of acid-resisting bronze.

For high lifts, especially with gritty water, a satisfactory pump valve is difficult to find. To reduce the unbalanced pressure on the valve at time of opening it is advisable to have narrow seats; this, however, brings high pressures on the seats when the valve is closed. On large valves for lifts of 600 to 1,000 ft. we have used leather, hard rubber, vulcanized fiber, and soft brass; for clear water the latter has given best satisfaction.

The difficulties due to grit make it advisable to have large sumps where water can settle, and if possible, to have two of them so that they can be cleaned alternately.

At the mines of the Old Dominion Copper Mining & Smelting Co., Globe, it was found that a small air ejector discharging into a mine car provides a very convenient way to clean mud and sand from a sump.

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The steam pumps can be divided into two broad classes: The direct acting, with simple, compound, or triple-expansion steam cylinders; and flywheel pumps with compound or triple-expansion steam cylinders. All the steam pumps are usually run condensing, both for steam economy and because exhaust steam cannot be discharged in the mine.

The steam consumption per water horsepower diminishes in the order given above and the complexity of the pumps increases as the steam consumption diminishes.

The simple direct-acting pump is so uneconomical in steam consumption that it should not be used for permanent pumping although very simple and convenient in emergencies. It can also be used with compressed air instead of steam.

The compound direct-acting pump for small power is a satisfactory pump.

The triple-expansion direct-acting pump is fairly economical in the use of steam if proportions of steam and water ends are correct for the lift. In a great many installations this is not the case and the steam end of the pump is too large for the work done, increasing considerably the steam consumption per horsepower. It is quite usual for mine managers to specify a higher lift for the pump than they expect to have when the pump is installed, so as to enable them to put the pump at a lower level later on; if a pump has to be ordered for such a change of lift it will be found advantageous to order the pump for the correct lift but specify the water end strong enough to stand the maximum pressure expected in the future. When lift on pump is increased the size of plunger can be altered at small cost to keep the correct proportions between steam and water end. This naturally decreases the capacity of the pump in proportion to the lift.

All the direct-acting pumps take comparatively little room and are easy to move from level to level.

The compound and triple-expansion flywheel pumps are more economical in use of steam, but are much more expensive of installation, require more room, and must be installed on good foundations that will not move or move as a whole. Generally their use can only be justified if the quantity of power to be delivered is considerable, steam expensive, and the probable life of the mine will repay the difference in cost.

Whether a surface condenser or a jet condenser should be used depends on local conditions. The surface condenser has less moving machinery and uses less steam than the jet condenser, but is liable to be out of service a longer time while tubes are being cleaned.

The question of proper size of steam line between boilers and pumps is one which usually is not given sufficient attention, most steam lines being too large for maximum economy. The writer has found in most cases that it is advantageous to have a steam pipe which gives a drop of pressure of 5 to 10 lb. between boilers and pumps, as the reduced con-

densation pays for the loss of pressure. When a small pipe line is used the usual separator at the pump should be made a separator-receiver, to steady the flow through the pipe line.

All steam pumps have a great advantage where flow is variable, as the speed and delivery of pump are easily adjustable to varying conditions.

The electric pumps can be divided in two broad classes: The plunger and centrifugal pumps.

The plunger pumps are of many designs; the writer generally prefers the vertical type as giving more even wear on packing and plungers and taking less floor space, although they require more head room.

For high lift the quintuplex pump having a practically steady flow on the discharge is advantageous, as it can be used without air chamber.

The motors are generally geared to the pump through a single or double reduction of gears. If straight spur gears are used it is not advisable to have a single reduction of more than 8 to 1, and slow-speed motors have to be used. If herringbone gears are used single reduction of larger ratio is permissible, allowing a motor of standard speed for pump drive. The herringbone gear is more efficient than the spur gear, but requires much closer adjustment of pitch line, so that bearings have to be made adjustable to maintain proper distance between pinion and gear shafts. For this style of drive the proper design of bearings is very important.

A flexible coupling between pinion and motor shaft usually pays for itself by reduced maintenance of motor, although on pumps using 50 hp. or less, a motor with outboard bearing, and either a fabric or raw-hide pinion geared directly to pump without coupling, has been found quite satisfactory.

With alternating-current motors, unless special motors are used, it is necessary to unload the pump at start by means of a bypass valve. Motors with high starting torque can be obtained, but are less efficient than standard motors, and are not necessary unless the pump is to be used with an automatic starter.

Direct-current motors will start the pump under full head, with a starting rheostat made for this service.

The writer has had little experience with centrifugal pumps. As a rule, however, they have the advantages of great simplicity, and ease of connection to motor, having no gearing; they are, however, less efficient than plunger pumps and have to be designed for exactly known conditions. They are sensitive to change of motor speed or changes of head, in a high-lift pump of moderate capacity the passages are small, the water travels at high speed, and if at all gritty the wear upsets the proportions of the pump and quickly affects the efficiency.

With plunger pumps the amount of water delivered can only be altered by varying the speed of the motor, starting and stopping the pump

as required by the flow of water, or by passing a portion of the water to the pump suction.

The first method is easy where direct current is available, but more complicated or less efficient where alternating-current motors are used.

The second method can be used with both systems of power, but is hard on the power plant if pumping load is a large proportion of the total.

The third one is inefficient in the use of power, as the pump is working at full load at all times.

As the efficiency of electric pumps is not much affected by their size, where flow of water is variable it is advisable to have more units of a smaller capacity so that the number of units running can be adjusted approximately to the quantity of water pumped and one unit started and stopped by hand or automatically to take care of the variation of water level in the sump.

Electric pumps usually require less attendance than steam pumps and reduce the quantity of heat liberated in the mine, which is a considerable advantage in hot mines.

The electrical measurements are so much easier to make than the steam measurements that an electric pump is likely to be kept at a higher efficiency. With an electric pump it is easy to show the attendants the evil effects of packing the glands too tight and the large amount of power that can be thus consumed without heating the plungers.

The electric plunger pumps can be operated at a lower lift than they are built for with very little loss of efficiency.

Where the proper submergence can be obtained an air lift is inexpensive to install and will handle very great quantities of water in a small space, and although the cost of the air is comparatively great, there are practically no other running costs. This method cannot be used from the lowest level of a mine, without a lot of complications, as there is then no way to get the proper submergence.

Air lifts were used at the mine of the Old Dominion Copper Mining & Smelting Co. at Globe in a recent emergency when water flow increased on the upper levels of the mine and got beyond the pumping capacity of the plant. A 10-in. air lift raising the water 200 ft. (exclusive of friction), using air at 90 to 95 lb. pressure at the power plant, required the following amount of air (measured by flow meter) at a barometric pressure of about 27 in. of mercury.

Gallons Water per Minute	Cubic Feet Free Air per Minute	Cubic Feet Free Air per 1,000 Gal.
1,011	1,353	1,338
1,680	1,809	1,080
1,794	2,262	1,261
1,925	2,658	1,375
1,965	3,219	1,638

Another lift raising the water 431 ft. exclusive of friction gave the following results:

Gallons Water per Minute	Cubic Feet Free Air per Minute	Cubic Feet Free Air per 1,000 Gal.
1,122	3,051	2,718
1,233	3,306	2,681
1,233	3,484	2,825
1,291	3,832	2,968
1,291	3,919	3,035
1,325	4,089	3,086

In these tests, which were kept up from 1 to 2 hr. each, the measurements of air were taken with a General Electric Co. air-flow recording meter installed carefully and checked after test according to instructions furnished by the makers; the results are probably correct within 5 per cent.

The measurements on water were obtained by taking the reduction of speed of the pumps when air lift was working and the water level in sump kept constant and a volumetric efficiency of 90 per cent. assumed for the pumps. All pumps were equipped with revolution counters.

The average speed of the pumps was recorded before and after the air test to insure that the water flow had not changed in the mine. The quantities of water are very nearly correct.

The above figures are not given as absolute, but only to give a rough idea of what can be done with an air lift. In both cases the submergence was about 190 ft.

In the following table are given the results of some tests on various sizes of steam and electric pumps. These tests were taken under running conditions and for steam pumps the water fed to boilers was taken as used by the pumps, all the auxiliaries of the boiler plant being run from another source of supply.

Tests 1, 2, and 3 were taken by condensing the exhaust of the pump and weighing the condensate and estimating the volumetric efficiency of water end at 90 per cent., as the delivered water was not measured.

Tests 4 to 9 were not taken by the writer.

Tests 10 and 11 are on triple-expansion four-cylinder pumps with two low-pressure cylinders. On these tests steam was superheated 35° when leaving boilers and the pumps were tested together.

Tests 12, 13, and 14 were on pumps immediately after starting.

Test 15 on same pump as No. 14 after two weeks' run.

Test 16 is on single-reduction spur-gear pump.

Test 17 is on single-reduction herringbone-gear pump.

Test 18 is the best test on air lift for 200 ft.

Test 19 is the next to the best test on same.

Test 20 is the best test on air lift for 431 ft.

Test 21 is the next to the best test on air lift for 431 ft.

On Tests 18 to 21 the steam consumption per water horsepower-hour is based on air compressors requiring 38 lb. of steam per 1,000 cu. ft. of free air, delivered compressed to 9 lb. gage pressure.

Steam Pumps

Test Num-ber	Kind of Pump	Steam Cylinders, Inches	Water Plunger, Inches	Stroke, Inches	Speed, Rev. per Min.	Water Del'd During Test, Gal. per Min.	Lift Including Friction, Feet	Steam per Water Horsepower-hour	Boiler Pressure, Pounds
1	Single Duplex.....	9	8 $\frac{1}{4}$	10	33.0	270	120	98.0	135
2	Single Duplex.....	14	8 $\frac{1}{2}$	10	33.0	285	111	135.0	135
3	No. 1 Exhausting in No. 2 as a Compound pump....	350	120	78.0	135
4	Triple Expansion Duplex....	10-15 $\frac{1}{2}$ -26	8	24	14.1	278	591	61.5	122
5	Triple Expansion Duplex....	15-23-39	10	24	12.7	391	926	49.4	125
6	Triple Expansion Duplex....	15-23-39	10	24	21.7	658	629	42.1	131
7	Triple Expansion Duplex....	18-28-47	12	24	13.0	584	1,056	35.9	143
8	Triple Expansion Duplex....	18-28-47	12	24	25.7	1,207	1,056	27.8	142
9	Cross-Compound Fly Wheel	19-35	5 $\frac{5}{8}$	36	32.4	502	1,059	23.7	110
10	Triple Expansion Fly Wheel	14-26-26-26	7 $\frac{7}{16}$	36	38.0	878	652	18.0	137
11	Triple Expansion Fly Wheel	14-26-26-26	6 $\frac{3}{8}$	36	32.0	553	883	18.0	137

Electric Pumps

	Kind of Pump	Motor, Horsepower	Water Plunger, Inches	Stroke, Inches	Speed, Rev. per Min.	Water Del'd During Test, Gal. per Min.	Lift Including Friction, Feet	Steam Based on 25 lb. per Kw-Hr.
12	Vertical Triplex Double Re- duction.....	10	4	9	35	51	475	32.7
13	Vertical Triplex Double Re- duction.....	20	4	9	37	54	1,162	28.1
14	Vertical Triplex Double Re- duction.....	50	9	10	37	302	423	29.1
15	Vertical Triplex Double Re- duction.....	50	9	10	37	303	423	25.6
16	Vertical Quintuplex Single Reduction.....	110	4½	12	40	159	1,155	23.1
17	Horizontal Quintuplex Single Reduction.....	250	9	18	52	1,245	605	22.2
				<i>Air Lifts</i>			Excluding Friction	
18	1,680	200	48.7
19	1,794	200	57.5
20	1,122	431	57.0
21	1,233	431	57.6

Fire-Fighting Methods at the Mountain View Mine, Butte, Mont.

BY C. L. BERRIEN,* BUTTE, MONT.

(San Francisco Meeting, September, 1915)

MANY fires have occurred in the mines of Butte in recent years, and while all have been of a serious nature, simply because they were mine fires, six of them have been especially dangerous in respect to loss of life and property, and the expense of extinguishing or controlling them. It is not my intention to discuss the relative merits of methods used in fighting these different fires because the conditions under which they occurred varied in each case, and it was not always advisable to expend the large sum of money necessary to actually extinguish them if they could be bulkheaded off from the rest of the mine. It is needless to say that everything possible has been done to check all fires at the beginning, but without success in some cases; and it has then become necessary to bulkhead all connections to them, in which state several now exist.

The mine fires of Butte, or fires in any mine, may be divided into two classes, namely: Those occurring in accessible workings, and those occurring in inaccessible workings; because these two conditions govern to the greatest extent the methods of fighting fires. Accessibility and inaccessibility in the case of mine fires concern not only new open workings and old closed workings, respectively, but also the condition of the ventilation in and about those places. When a fire is accessible at the start it should not be difficult to confine it to a small area and extinguish it in a relatively short time, but when it is impossible to actually reach a fire in old filled, abandoned workings, or there is no opening by which the gases may be conducted away from the fire zone, then the time consumed in changing those conditions permits the fire to increase to such an extent that it is a long, hard fight to overcome it.

These different conditions have been met in Butte, and of the six fires of serious proportions the Steward mine fire and the High Ore-Modoc mine fire were in new accessible workings and were extinguished in a month or so. The Anaconda mine fire, the Leonard-Minnie Healy mine fire, the West Colusa mine fire, and the Mountain View mine fire, were all inaccessible. The Anaconda mine fire started about 25 years ago and though safely bulkheaded is still active. The Leonard-Minnie Healy

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fire is about eight years old, bulkheaded and active. The West Colusa fire began about two years ago, is now bulkheaded, but is still alive. The Mountain View fire originated in June, 1913, and at the end of December, 1913, to all appearances was extinguished.

I had been at the Mountain View mine for three years as assistant foreman, and previous to that was in the geological department about six years. In that time my work took me to many different mines of the company, and owing to the existence of other mine fires the smelling sense in regard to the resultant gases had become well educated.

Aside from that, I had always noticed that each mine, or different places in the same mine, had a distinctive, though possibly faint, difference in odor. At any rate, a man knows or should know the odors of the mine in which he works and investigate the slightest change in them. Of course any one can notice decided changes, but it is the first faint odors to which I refer, and the reason I mention this as being especially important is because a month or so may be gained or lost in starting work toward the source of danger when the fire is not an open one. Furthermore, the fault of not recognizing these things immediately, or of not thoroughly investigating the slightest suspicion, may lead to loss of life, or something less serious, as in the case of the West Colusa fire, when gases from there came into the Mountain View mine on night shift and bosses failed to report it because they were not positive. As a result 30 men on the day shift were overcome when they went to their working places.

I say the Mountain View fire originated in June, but by that statement is meant that at that time we actually knew a fire existed. It is a fact that for some weeks previous to that date we had our suspicions that something foreign was in the air of the mine which could not be traced to any definite location. On two successive change days when only repair men were working and the air of the mine was less disturbed in places, because no air drills or cages were running in the mine, the writer detected a faint odor of gas which could not be traced then and which disappeared entirely when the mine was working. On June 18, 1913, several shift bosses of the day shift and the mine foreman had detected the smell of wood smoke, but not being positive asked the writer, then on night shift, to investigate.

They thought the trouble was near 550 crosscut, the location of which is shown on Plate 8, so I went there first and at the top of B-614 raise could smell wood smoke. Going down the raise 10 ft. and west 50 ft., along the top of the stope filling, a haze of wood smoke was evident. This stope had been worked out and filled all but 10 ft. under the 500-level sill six months or so before. West of the cross mark was a solid gob of waste and square-set timbers 12 ft. wide, six years old, and the smoke was coming from below and west of that point. The smoke was not thick enough at that time to harm a person, but was visible and sta-

tionary, there being no movement to it on account of everything being filled below and west.

We had a fire, and the first move was for the safety of the men, and the next for water. The ventilation of this part of the mine was by air traveling up, and in its course was passing 30 men working in east-end stopes from the 500 to the surface. The shift boss on that run was sent to bring those men to the 500 while fire hose was brought from the surface, attached to the water pipe in the main air shaft and run to the stope at the cross. It took about 20 min. to do this and inform officials of conditions. There was no fire, smoke, or gas on the 600 in drifts directly under the cross nor were there any other workings open on the 500 by which we could get nearer the cross aside from A-557 drift and 550 crosscut. All workings south of A-557 and west of 550 as far as Fan 3 were caved or filled. (See Plate 8. Solid lines on level maps show workings before the fire. Dotted lines show work done after fire started.)

Feeling positive that the fire was west of the cross, the men were removed from the stopes above and put to work immediately to clean out A-581 and 586 in order to get water to the stopes under 515 drift. At the same time similar work on the 400 level was started at A-445 and 412 off 449, and at A-423 and A-466 off A-462, to get water into stopes under 416, the old hanging-wall drift. (See Plate 5.)

The first work the following day was to change the entire ventilation on the east end of the mine so that we could draw the gases out of the mine and continue to operate. We had the two-compartment Sullivan air raise, each compartment being 5 ft. square in the clear, on the east end of the mine, which at that time was downcast.

At the collar of this raise was installed a No. 15 Sirocco fan changing it to a strong upcast, which later enabled us to work toward the fire from the main part of the mine. For eight days we worked along these lines, getting water as close to the source of the fire as possible, but instead of the smoke decreasing it gradually increased until we could not go below the sill of 550 crosscut. During that time, however, we had put in a 2-in. pipe line for water on the 400 level from the main air shaft to A-476 crosscut by way of A-479, A-409, 495, A-452, 449, A-462, and 441, with tees at each crosscut leading to the south. On the 500 we had put in a 2-in. line from the main air shaft to the junction of 550 crosscut and A-576 drift by way of 591, 594, A-583, A-557, and 550, with tees at each crosscut. These pipe lines were hung to the timbers in the drifts. We had also put in a 2-in. line on the 600 from the main air shaft to a point under the fire on the 500 by way of A-622, A-645, and B-642, where we kept men all the time to watch for fire from above. These lines were all connected to a 2-in. water line in the shaft coming from the city water line on the surface. The pipe was made to stand a pressure of 440 lb. The 600 level is 887 ft. below the collar.

All the time we had Draeger helmets of the 1904-09 and 1910-11 types ready for use, and a man on each shift to keep them repaired. We had all had practical experience in using them.

On the ninth day of the fire the writer took charge of the mine. Conditions were growing worse each day and we realized the necessity of starting in systematically to protect the rest of the mine and getting water through all the stopes from the top to the 600. It was evident at the time that the fire would spread east and west and to the levels above and below before we could cut it off.

We decided to run lateral drifts east along the old south drifts on the 200, 300, 400, and 500, and directly over the stopes below, keeping the old timbers in sight on the left of the lateral and solid ground on the right. We further planned to put water into the old workings as we advanced and at certain intervals to run crosscuts south where we could set up diamond drills to drill holes to the stopes below. We knew we must get water into the stopes elsewhere than directly from the levels because some of the stopes were so wide and flat that the water from levels would only cover the foot wall. To carry out these plans we needed air pressure to force the gas and smoke ahead of us, and lots of water.

A new 4-in. pipe for water was put in the main air shaft and with the 2-in. column already there we had enough water to supply five 2-in. lines on the levels at full capacity. One line was run on the 200 level to the Sullivan upcast, one line on the 300, a new line on the 400, where we already had one, a new one on the 500, where we already had one, and with the one on the 600 gave us seven 2-in. lines, which were brought along as we opened new country in the fire zone. We also installed a No. 4 Sirocco fan on the 400 level at the junction of A-409 and 495, one at position marked fan No. 1 on the 500 level, which later was moved to position of fan No. 2, and one at position marked fan No. 3 on the 500 level. The air was conducted from these fans by galvanized-iron pipe 10 in. in diameter, where it was to serve one place, and by 18-in. pipe branched to 10-in. where the fan was to serve more than one place.

To maintain the air pressure in the working places we always built two bulkheads 15 ft. apart so that one could always be closed while men were going and coming. These bulkheads consisted of a tunnel set of 10 by 10 in. posts 7 ft. long with only a 3-in. or 5-in. cap; the space between the set and the ground was concreted.

The galvanized pipe, air pipe, water pipes, an extra pipe for emergency, and electric-light wire pipe rested on the cap and were concreted in place. The doors were of wood and opened inward so they would always close with the pressure. A flap of canvas was nailed on the bottom of each door to prevent the air from escaping. Aside from these working doors a solid concrete bulkhead with sheet-iron doors opening outward was built in each drift for safety in case the fire drove us back.

Speed and safety were the principal objects in the work and the less ground we had to move the faster we went.

Before describing the details of work performed it would be well to mention conditions in general from the tenth to the twentieth day. The mine was producing at that time 1,600 tons a day and throughout the fire period never missed a shift. We were then running 200 gal. of water a minute on to the fire, which after the first month was increased to 670 gal. per minute. This rate was kept up for five months and the water carried about 1 per cent. copper when it reached the pumps.

Bulkheads, dams, and water boxes had to be put in on levels below the 600 to the 1,400 to handle the water coming from the fire zone, and this with the repair work gave us a great deal of trouble. On the eleventh day, fire was discovered at the cross on the 600 level (see Plate 10) in the timbers over the drift and along the back to A-696 drift, making 20 ft. of fire. The ground above was stoped and filled over A-693, A-696, 623, A-635, and A-655, and gave an excellent opportunity for fire to descend. Water was then coming down to the 600 through some of these workings. The only open drifts under the fire on the 600 were B-645, B-642, A-696, A-690, and B-616.

Looking at Plate 11, which shows the workings one floor under the 600 level, one can see the possibilities of a fire to pass the 600 and go on down through the mine. For five days we had an interesting time keeping the fire above the level and it was necessary finally to have two hose lines on the work there with six men on each 6-hr. shift. Aside from that it kept four men busy all through the fire period keeping open the drifts, which were very heavy on account of water coming from above. For several months, until we ran a new lateral drift south of the old workings, a man going in through B-642 to the east could come out only the way he went in, and with the smoke and fire along the back the men on the 600 had a far from pleasant time. Fearing that the fire would drive out the hose men we laid lines of perforated 2-in. pipe, wrapped in tarred canvas, along the bottom of all the open drifts, our intention being to run water through them should access to that country be cut off.

Because the fire and the work on it were assuming such proportions, it was decided to separate them from the regular mine work; and mine fire bosses, three on each 8-hr. shift, were appointed. The assistant foreman and regular mine bosses took care of the ordinary operations in the mine.

On the 500 we had reached 523 drift in A-581 crosscut and had installed diamond drill No. 1, with which we had drilled several holes, from all of which the core barrel came out almost red hot. (See Plate 9.) Gas and sulphur smoke were plentiful and we were working under pressure from fan No. 1. These fans with the air discharging inside our double

doors kept back the gas and smoke in nearly all cases, but when they failed we used the compressed-air system in conjunction with them.

Access to 550 crosscut was possible with Draeger helmets only. Water from the 400 was hindering the work considerably. On the 400 level we had run B-430, A-445, 412, A-423, and A-466 crosscuts in over the stopes and were putting down water at those points. In these places the men wore hoods on the back of which a small hose was connected for the use of compressed air. On the inside of the hood, over the connection, a small square of canvas was attached by its four corners to prevent the air from blowing directly on the head and to send the air around the face. These hoods were used in much of the work where a Draeger helmet was not needed. Carbon monoxide and dioxide were present on the upper levels from the beginning of the fire.

On the 300, during this first 20-day period, the gases appeared only on the east end of the mine on their way to the surface through the Sullivan raise.

A dam was built near the junction of 319 and B-314 over which water was run continuously. Nearly all workings east of that point were inaccessible because of being broken down or full of gas.

On the 200 up to that time we could go into the Sullivan raise, where for some weeks we judged conditions below by the smoke and gas passing through it. We had water piped to it but did not turn any in until some time later, as we feared we would weaken the raise and lose our upcast.

Along with our general plan of hanging-wall laterals and diamond-drill holes, as a means of quenching the fire entirely, we decided to build a fireproof wall west of the fire zone from the 200 to the 600 and a fireproof floor on the 600 under the fire to save the mine west of and below those boundaries should we fail to control the fire. This work was started by raises from the 300, 400, 500, and 600 levels, so arranged that they made an almost continuous raise from the 600 level up. The details of this work will be explained later.

Throughout the fire period all of the work accomplished was carried on with some interruptions and many difficulties due to the increasing area of the fire and gases.

An account of all operations on different levels after the first 20 days will now be given.

On the 200 level (see Plate 1) for the first month we had access to the Sullivan raise, but about that time the ground around the top of A-322 raise caved, due to timbers burning out below, and left a hole 20 ft. wide over which we could not pass. The hole was bridged with three 15-ft., 10 in. by 10 in. stringers tailed one upon the other and the hole filled with waste and concrete. All the work was done by men working in Draeger helmets and was the hardest work of that kind we had.

The vein in 266 drift had been worked out from the 300 to the oxidized zone three floors above the 200, and west to within 10 ft. of 259 under the 200. Above the 200 the stopes, two floors high, ran west of 259 and over 296. At the junction of 259 and 266 we built a concrete bulkhead 35 ft. long, 20 ft. high and 4 ft. thick. This work was done in Draeger helmets, and on the top 5 ft. the men had to wear heavy gloves on account of the heat; it took two months to build the wall but we had the drift free from gas and smoke. By that time we could only get to within 60 ft. of the Sullivan raise because the drift had caved for that distance.

Fire had now found its way up to the 200 in the Sullivan raise, and it had become necessary to stop the fan and turn water down the raise from the surface or lose the entire raise. The fan was not run after that, as we were able to keep the gases back with our underground fans. We had cut off all unnecessary air from the fire zone below and on the 200 some of the downcast air from the main air shaft was allowed to travel east and up the Sullivan, thus holding back the gases on the fire below. As a result, we maintained a condition of rest for the air in the fire zone, which in my opinion was one of the greatest aids in the work. We could open the doors on the 200 level 6 in. more than required to maintain these conditions and force the gas back against the fans on the 400 and 500 levels. The miners on the 200 often demonstrated this for us by their carelessness in passing in and out.

A lateral drift, A-254, was run 20 to 30 ft. south of 266 and from it crosscuts were driven every 15 ft. to within 5 ft. of the old workings. Into the face of each crosscut two holes were drilled to pierce the old workings, and through each hole 1-in. pipe for water was run. This gave us 14 crosscuts and 28 water pipes, from which we soaked the stopes below. (See Plate 1.) Dotted lines show workings run for this object. Crosscut A-275 was run to discharge water into the stope below 259 drift.

Just east of the main air shaft on the 200 may be seen the top of fire-protection raise B-385, which was run up through the old gobs. From there A-268 crosscut was run south, and at the end a new air raise was run 450 ft. to the surface to be used for ventilation after the fire.

On the 300 two lateral drifts, B-390 and B-381, were run from B-353 and A-380 respectively to pass over the stopes below. (See Plate 2.) Dotted lines indicate the new work on fire.

We followed the old filled drifts where it was possible and ran water into them as we advanced. When A-368 was reached we found water coming down from the 200 and decided it would be unnecessary work to advance farther. Near the south end of B-383 crosscut we put up fire-protection raise B-385 through old gob and farther north in the same crosscut, but not showing on the map, we ran another raise to the 200 through old gob.

The work on the 400 level was more extensive. (See Plate 5 for loca-

tion of fire drifts, diamond drills, and fan.) The dotted lines show work done to reach the fire zone.

As stated, a Sirocco fan had been set up as shown with galvanized pipe leading into B-419 and around to A-445. Crosscuts B-430, A-445, 412, A-423, and A-466 had been run to the old stopes. We ran B-419 in about 100 ft., crossed the old drift, 416, and from there continued east over the old stopes below until opposite A-466 crosscut. In one month, 315 ft. were run and timbered by four 6-hr. shifts with five men on a shift. Two men drilled with a cross bar, one shoveled, and two ran car. B-422, B-425, B-426, B-421, and B-423 were run at the same time the drift was advancing.

Water was put down in all places as fast as it would sink away. Our belief was that the fire would beat us to the 400, and so it proved, for at the end of B-419 we ran into a mass of fire. A-462 at that time was filled with dense white smoke in which investigation work with Draeger helmets was very dangerous.

All of these workings were difficult to maintain on account of water from above.

Diamond drills were set up as indicated on Plate 5. The courses and dips of all these holes were set with a Brunton compass by the writer. The position of the drills and the starting points of the extreme holes of each series of holes were located before the drills were set up. Having these accurately located the intermediate holes were easily placed so that the country below would receive water at every 10 ft. From these three setups the holes shown on Plate 6 and Plate 7 were drilled. From diamond drill No. 3 the upper holes on Plate 3 and Plate 4 were drilled; also from drill No. 3, a series of holes was drilled to the 500 sill along 515 drift.

As soon as each hole was finished it was piped and water turned in immediately. One of the hardest things to do was the piping of some of these holes because of the gases which came up through them.

It is impossible for one to realize what a busy place B-419 drift was during these times and especially during the installation of apparatus, etc. All the pipes had to be wrapped with tarred canvas to protect them from the strong copper water; the rails would become eaten out; the side lagging break in, and other innumerable things happen to delay the work.

In between times, we had trips to make in Draeger helmets to different places, to note conditions, etc., through drifts a foot or two deep with water and other obstructions, or filled with sulphur smoke.

We ran B-427 just east of drill No. 3, from which place B-428 and A-488 crosscuts were driven. All the timber had been burned out there and the ground was so hot that it took 15 min. to cool the drill holes for safety in blasting. In B-428 we had the only accident during the fire.

One of the upper holes from diamond drill No. 3 had holed to the stope above and the water from it, turning to steam, caused an explosion that blew hot dust down into the crosscuts. Two men working in B-428 were very badly burned.

Near the fan, fire-protection raises B-431 and B-432 were run up through the old gobs.

On the 500, we had reached 515 drift in A-581 and had drilled six holes from diamond drill No. 1 as shown on Plate 9. The fire spread so fast that we lost the drill and all the rods and were driven out completely. Fan No. 1 was moved back to fan No. 2, where we built large concrete bulkheads with iron doors, beyond which we could not go except with Draeger helmets.

Work on the 500 was then confined to running 515, B-536, B-538, and B-540 as shown by dotted lines on Plate 8.

Diamond drill No. 3 was set up and the holes as shown on Plate 9 were drilled. The stope was full of fire and very difficult to extinguish. Many times we were driven out of 515 by the steam and smoke caused by water running from the drill holes on the fire.

We were very much concerned at this point because the fire was working west between the 500 and 600. Drift 515 was being extended all this time, the face of which was in burning timber a long way before our water from above hit it. In the last 100 ft. of 515 we ran into many of our water pipes which we had run down in drill holes from the 400.

Diamond drill No. 2 was set up in B-548 crosscut, which was run especially for the purpose from a south drift, it not being thought safe to place it anywhere along 515. The dots on Plate 9 indicate the holes drilled. We had intended to run a row of holes to the fifth floor of these stopes but this was unnecessary.

During the worst period on the 500 we decided to try steam, so a 3-in. line was put down the shaft and branched to two 2-in. lines; one of which discharged east of the bulkhead by No. 2 fan and one inside the bulkhead in B-535 near No. 1 diamond drill. From observations, made in Draeger helmets, we decided the steam did not help much and so gave it up.

At fan No. 3 and 50 ft. north of it, we put up two fire-protection raises through the gobs to the 400.

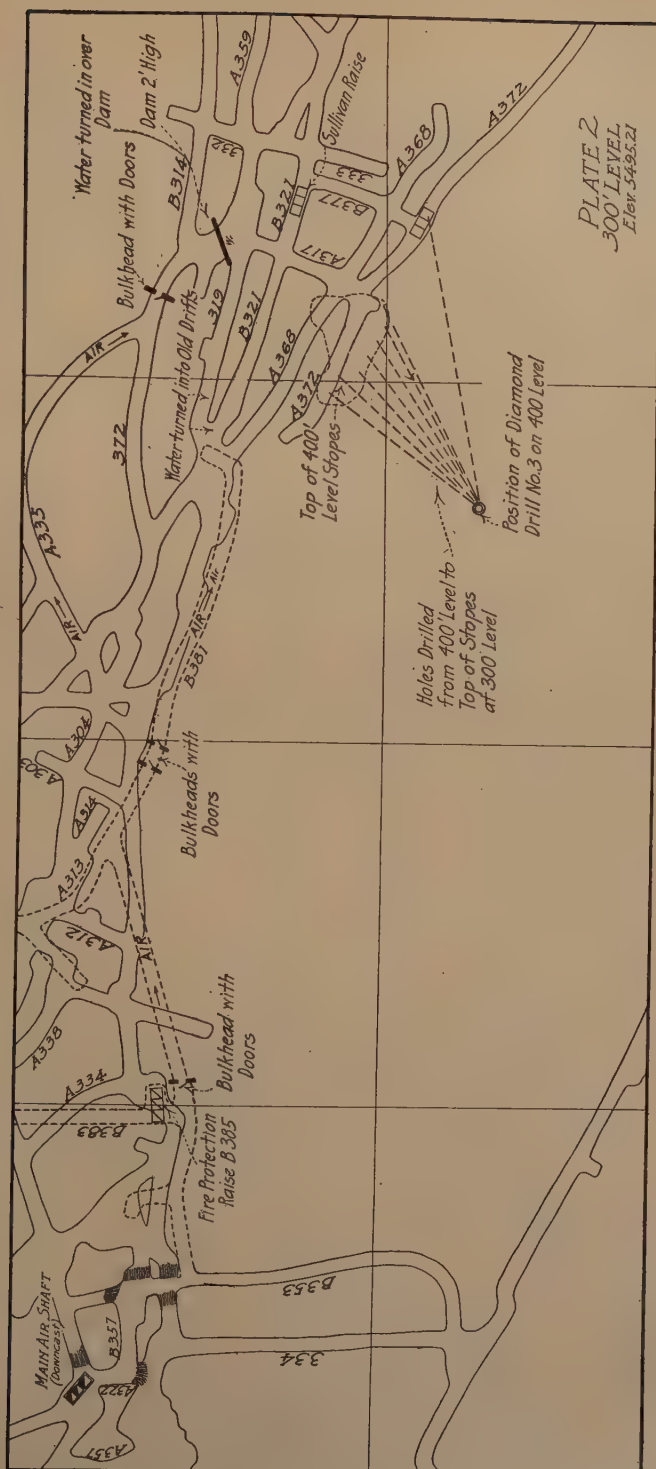
The work we had planned for the 600 sill seemed endless when we started, as water was coming down in such volume. Plate 11 shows the stopes and raises which reached the 600 from the 700. The dotted lines show the drifts which were run during the fire to reach these places. The plan, as stated before, was to make a fireproof floor 8 ft. thick by removing all timbers in the stopes, drifts, and raises, on the 600, and filling in with waste and concrete.

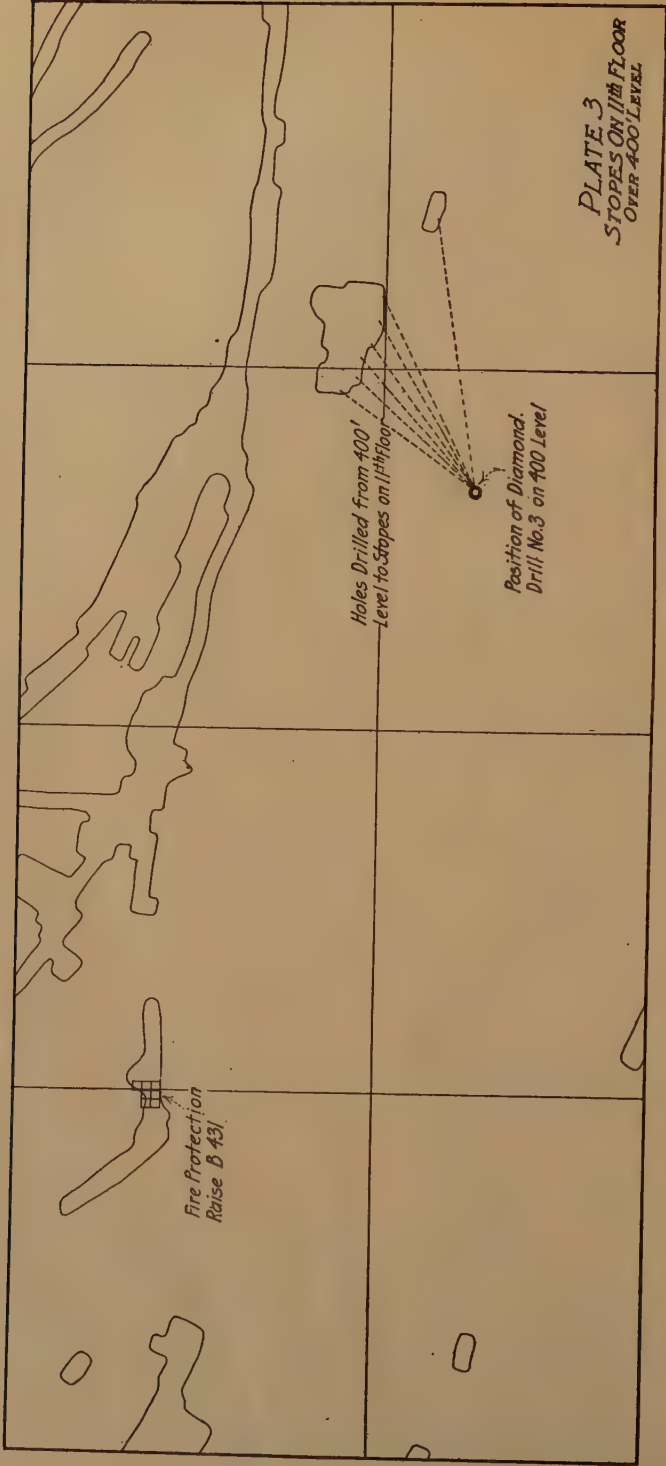
The first step was to run these new drifts through the old broken down

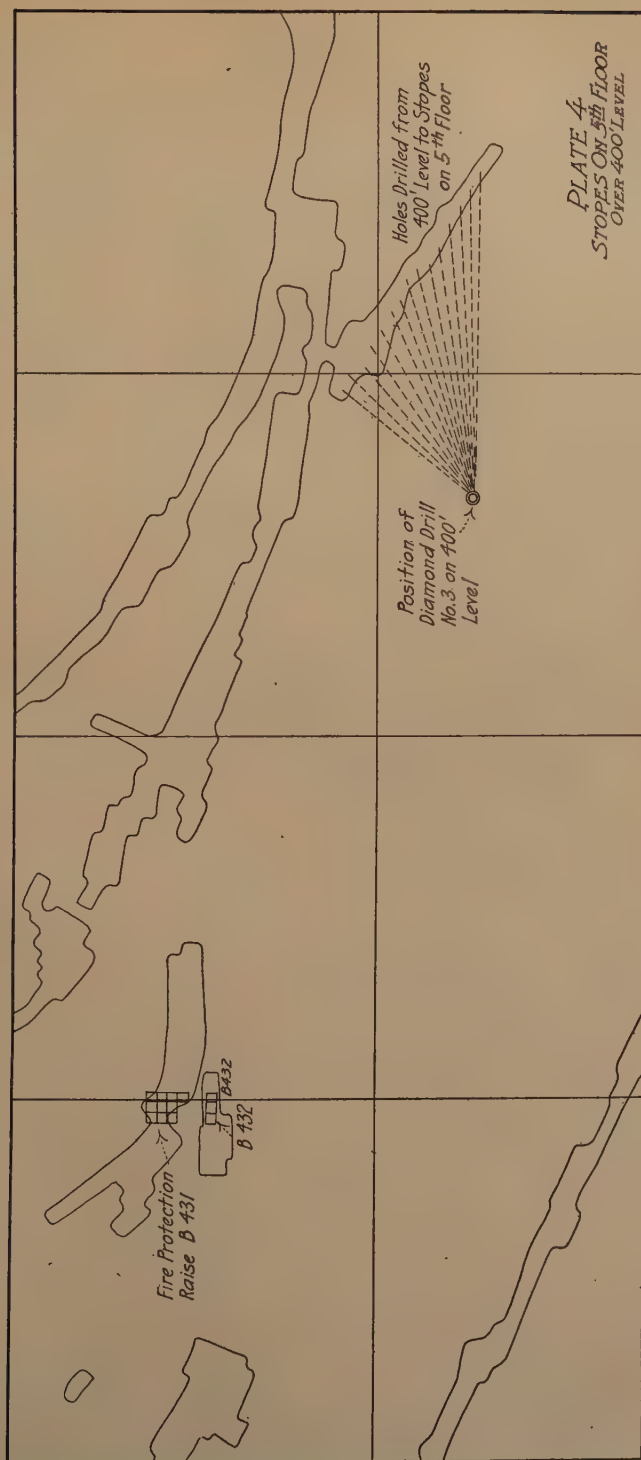
places, putting in new timber as we advanced. After an old drift had been opened in this way, we started at the end and came back, removing the wood posts and putting in old air pipes in their places, then filling the whole drift with waste. The work was arranged so that the waste from places we were timbering was banked in the places we were filling; 1,500 ft. of this was done, all by contract. The cost was high because: The original work necessitated spiling; 670 gal. of water a minute were running through these workings; the air was so poor that candles were useless and carbide and electric lights had to be used; and much time was lost in repairs to the tracks and pipe lines. For some time hot water made it impossible to work in some places. Some of the raises were concreted for a set below the sill after the timber had been removed. All the concrete was mixed by machine at the collar of the shaft. Plates 10 and 11 give a good idea of the extent of the work.

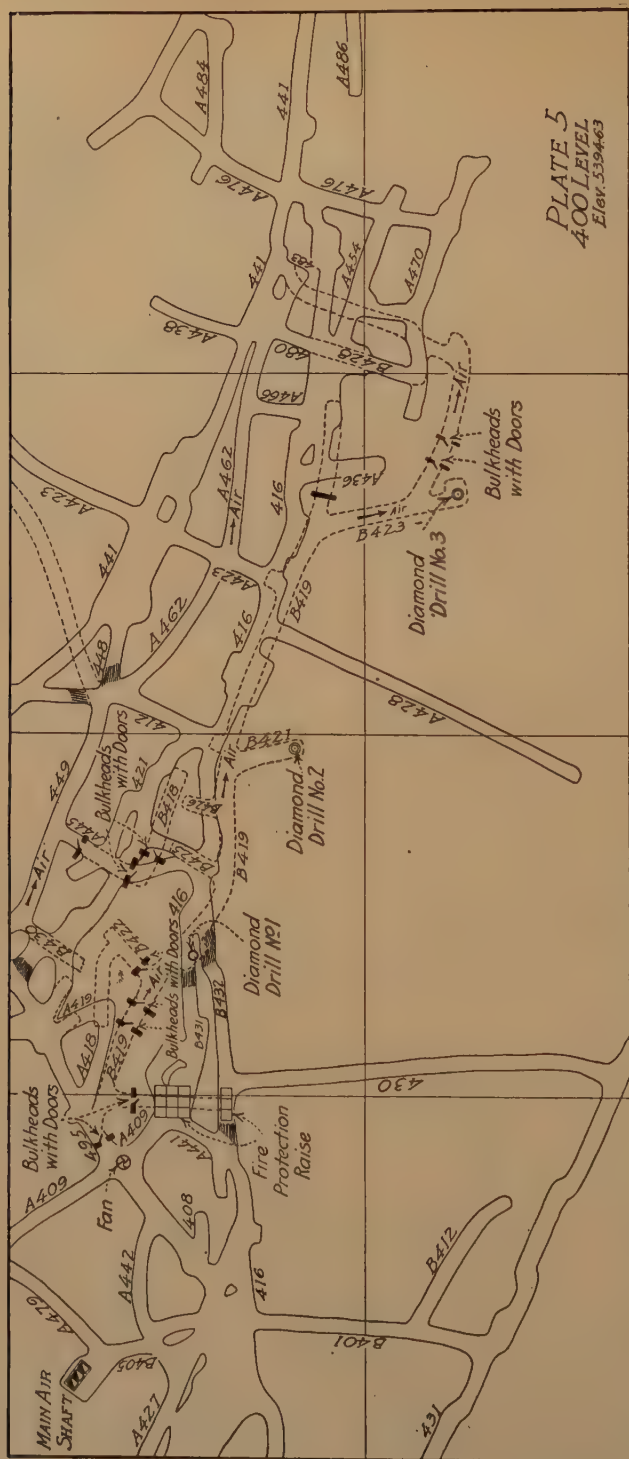
Fire-protection raises B-650 and B-651 were run through the gobs above to the 500 level. (See Plates 9 and 10.) All of the raises from the 600 to the 200 were three sets long east and west, only the two west sets of which were timbered. The west set was the manway, the center set was the chute, and the east set was filled with waste. When all were finished we had a wall of dirt filling from the 600 to the 200 which was fireproof but not gasproof. This we decided was sufficient if we should extinguish the fire. Had it been necessary, we decided to fill the chutes with concrete to make them gas and fire proof. Near the raises on each level we built permanent bulkheads with iron doors to cut off the fire zone from the rest of the mine.

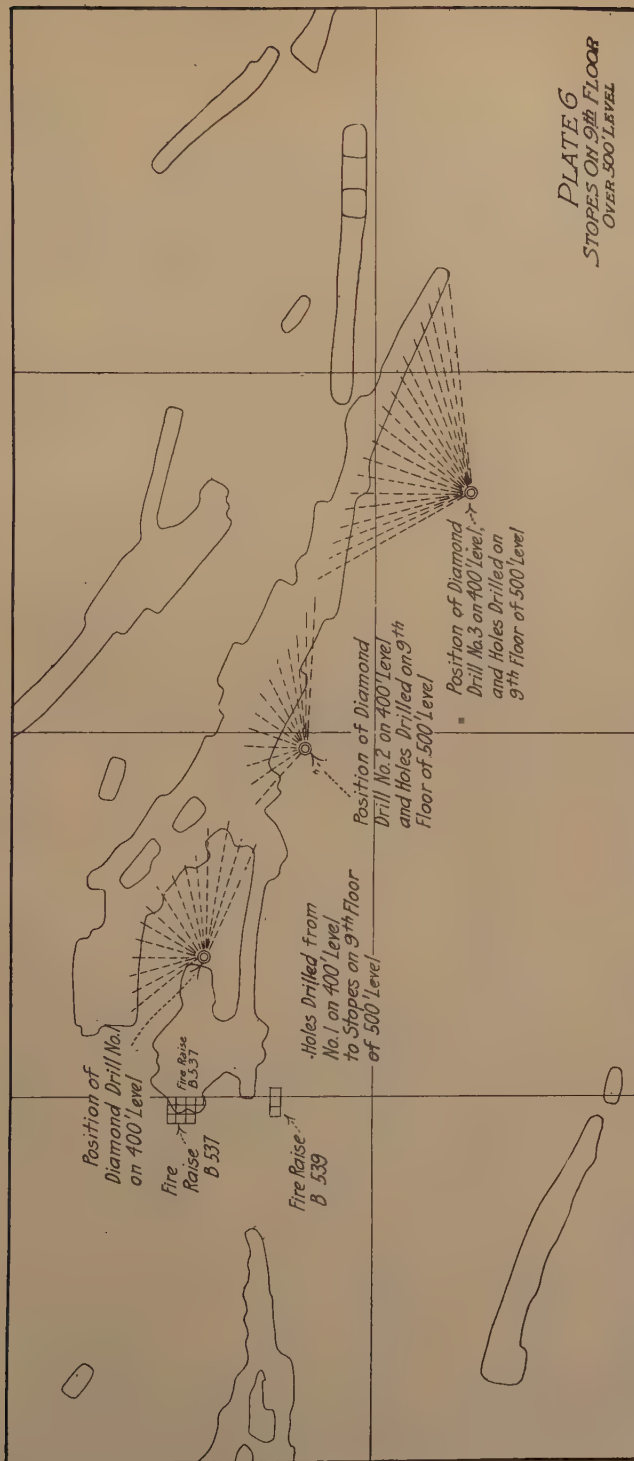
At the end of December, 1913, or six months after the fire started, we could say positively that there was no fire in the mine. As late as Dec. 1, 1914, this fire zone has been inspected and no trace of gas or smoke was evident.

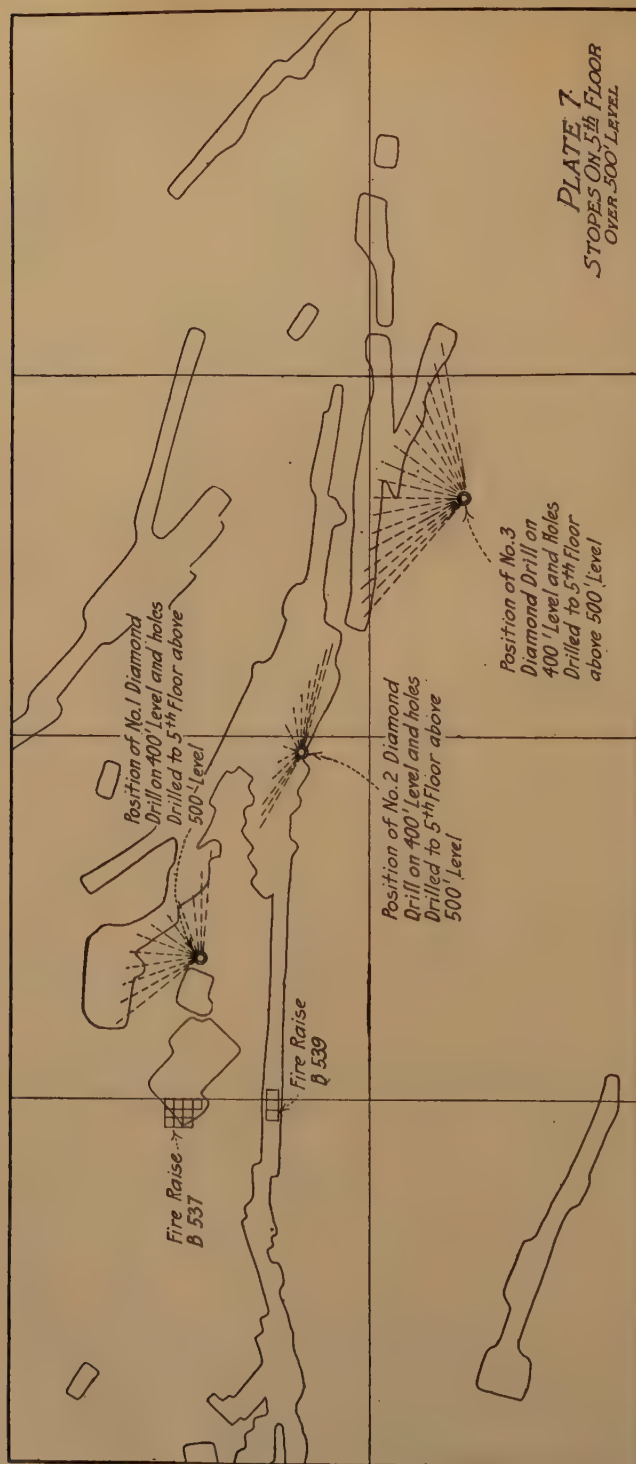




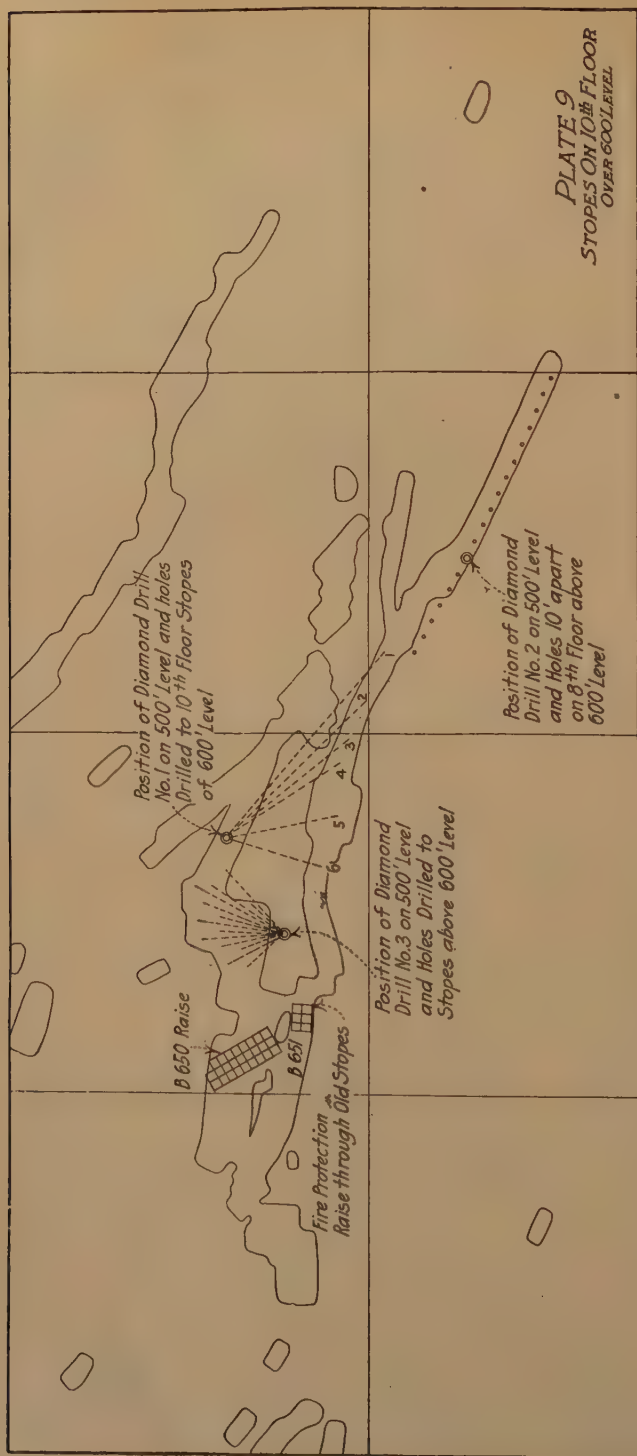


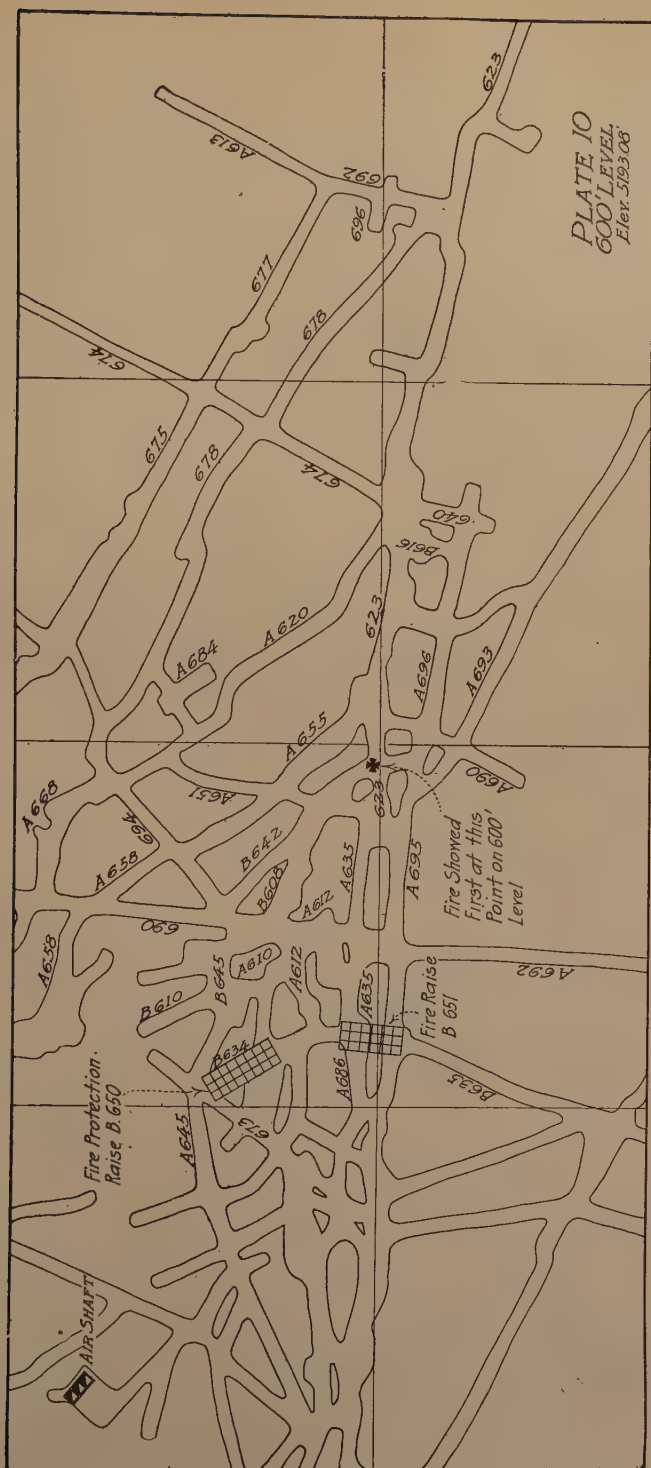


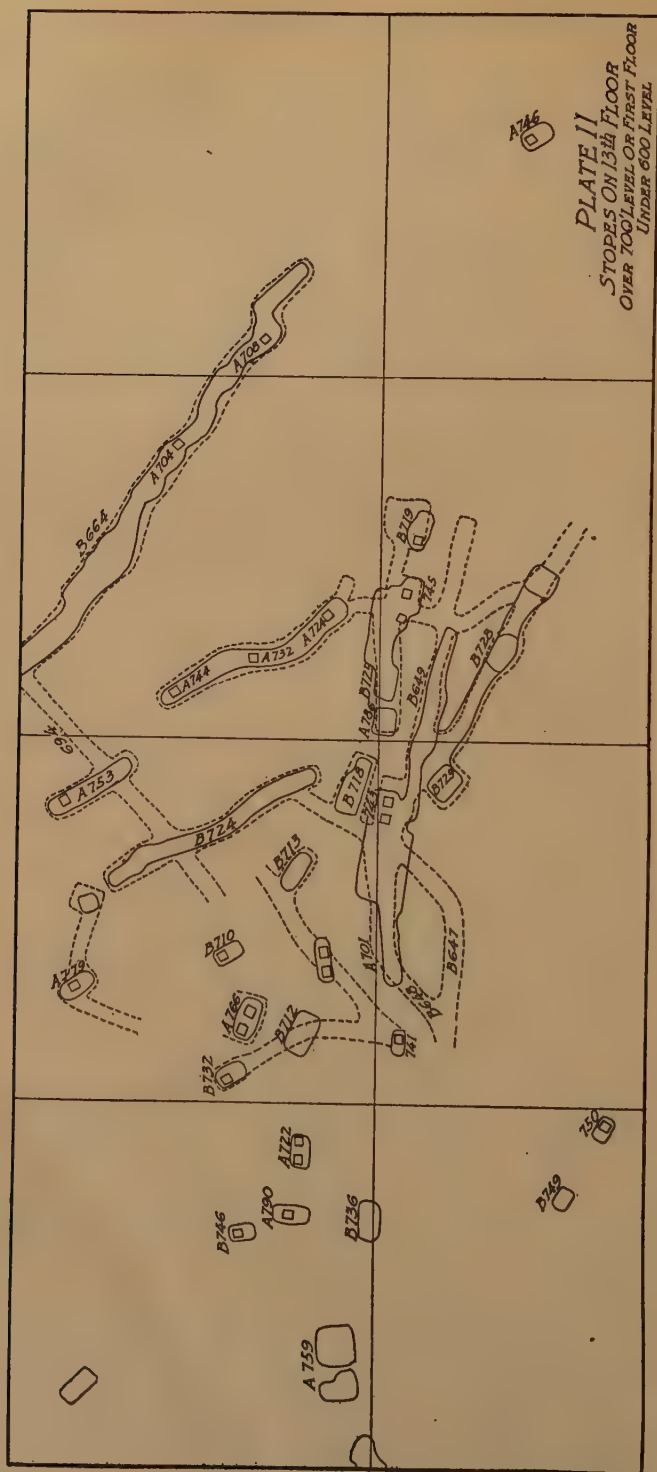












The Application of the Apex Law at Wardner, Idaho

BY FRED T. GREENE, E. M., BUTTE, MONT.

(San Francisco Meeting, September, 1915)

MOST of the recent discussion of the mineral land law published in the *Transactions* is in the abstract—an exception being Mr. Goodale's paper, *The Apex Law in the Drumlunnon Controversy*,¹ which illustrates in concrete instances many of the practical disadvantages of what Dr. Raymond has called "The Law of the Apex." The present paper, it is hoped, may serve a similar purpose, emphasizing the necessity for a change in our mining laws, and bringing home to mining engineers some of the defects and dangers of our present code.

There are many storm centers of "apex litigation" in the Rocky Mountains; and not the least of them is the Wardner camp, where such litigation has been almost continuous since 1892, and the end is not yet.

In their decisions, the courts have accepted the claims of the litigants that the "Wardner vein" is a broad zone varying from 250 to 500 ft. in width at the surface. This is the simplest theory which could have been advanced, and probably decreased, rather than augmented, the number of contests; in fact, every one of the claim owners seemed to feel that the greatest advantage would accrue to him if this theory were adhered to. The dominant structural feature of the Wardner vein is a persistent fault, striking N. 40° W., dipping about 40° to the southwest and called the "Wardner foot wall." Fig. 1 is a composite of two adjacent cross-sections of the vein and is a fair representation of the relative size and shape of the orebodies and their relation to the "Wardner foot wall." The country rock is a comparatively thin-bedded sericitic quartzite which has an average strike of N. 80° W. and dips at high angles to the north and south.

The movement on the Wardner foot wall is apparently normal, and there has been more or less brecciation on either side of it; but as the moving mass was that above the foot wall, the brecciation has been more complete on that side, and has extended farther from the plane of movement. Many minor faults, generally of insignificant extent, resulted from the movement along the Wardner foot wall. Most of these were

¹ *Trans.*, xlviii, 328 to 341 (1914).

pre-mineral, although some post-mineral dislocations can be observed. The period of mineralization must have been very long, and these minor faults and the bedding of the quartzite largely controlled the shape and position of the orebodies. The average strike of these minor faults is N. 14° W. with a predominating dip of about 60° to the west. The ore seams have an average strike of approximately N. 50° W., which is practically a mean between the bedding planes and the minor faults.

The orebodies rarely rest immediately upon the Wardner foot wall,

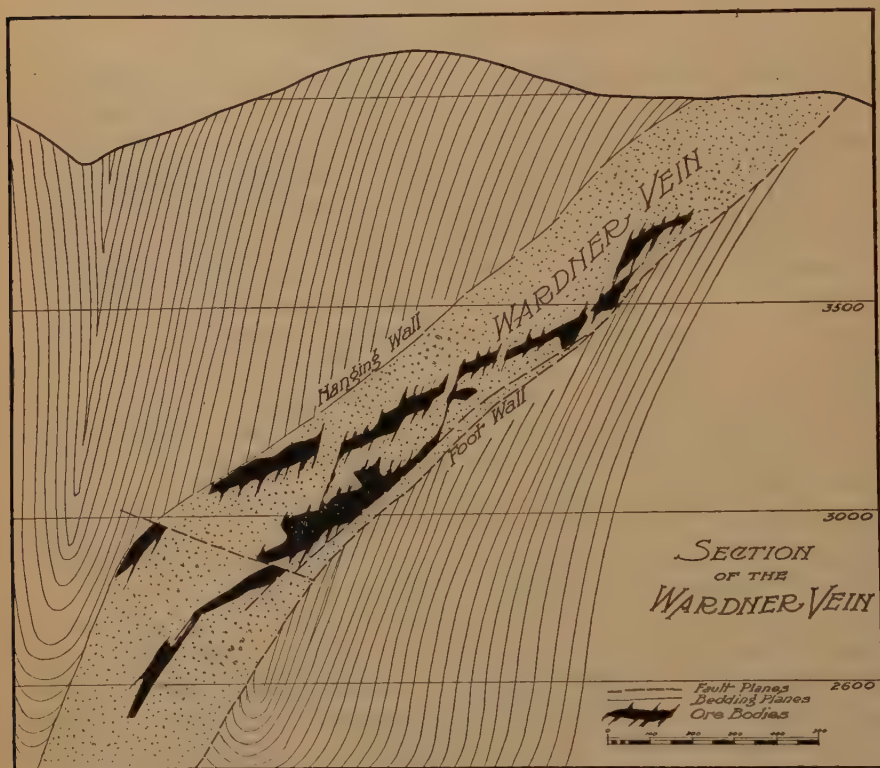


FIG. 1.—SECTION OF THE WARDNER VEIN.

but depart from it at sharply acute angles both along the strike and dip. By reason of extensive alteration and oxidation along bedding planes, this departure of the orebodies from the Wardner foot wall was even more pronounced at the surface; so that it is not at all surprising that the discoverers, in the necessary hurry of marking their boundaries, placed their claims more nearly parallel to the bedding than to the foot wall. It is to this error that most of the litigation at Wardner has been due.

There are four fundamental conclusions of law regarding the extra-lateral right of the discovery vein:

1. When the vein passes through both end lines. This condition is, of course, covered by the text of the law.

2. When the apex of the vein crosses one end line and stops before reaching the other. Here again the end lines define the right.²

3. When the apex enters the claim through one end line and departs across either side line. In this case the extra-lateral right is defined by two planes, one through that end line, and the other through the point of departure across the side line, and parallel to the end lines.³

4. When the apex of the vein crosses both side lines. Here, providing the side lines are parallel or converge in the direction of the dip, the side lines become end lines and the extra-lateral right is contained between vertical planes through these side lines extended in their own direction.⁴

This feature of the apex law has been analyzed by Dr. Raymond,⁵ who introduces his discussion of it by remarking:

"There is apparently no end to the doubts, inconsistencies and absurdities in which the courts of our mining States and Territories are involved in their attempts to apply to conditions of ever-increasing complexity the provisions of our anomalous mining law."

This language was prophetic and would be none the less so were it used today.

In no mining camp has this question of side-line crossings been more frequently raised than at Wardner. Had individual claims been maintained, the accompanying sketch, Fig. 2, based on the conclusions of one of the ablest mining attorneys in the Rocky Mountain States, would define the ownerships in that camp. Had not the controversy between the two companies operating along this vein been settled out of court, it is easy to conceive that the litigation over extra-lateral rights would have been interminable.

Twenty years of litigation established the rights of the following claims as shown in Fig. 2: Lackawana, Sullivan, Bunker Hill, Stemwinder, Emma, Last Chance, Tyler, King Fraction, Viola, and San Carlos; but the rights of the Phil Sheridan, Republican Fraction, Cheyenne, Lincoln, Sims Fraction, and Overlap, remained to be determined.

The Phil Sheridan right would probably be settled more or less automatically; but, because of the immense orebodies between the Phil Sheridan and Tyler extra-lateral right, suits based on the Republican Fraction, Sims Fraction, and Overlap, would have been bitterly contested.

An early effort to acquire the Republican Fraction right was fruitless; but the "doubts, inconsistencies and absurdities" of the apex law are so

² Carson Co. vs. North Star Co., 73 Fed., p. 597.

³ Fitzgerald Co. vs. Last Chance, 171 U. S., p. 55.

⁴ Flagstaff Silver Mining Co. vs. Tarbel, 98 U. S., p. 463; Argentine Co. vs. Terrible Co., 122 U. S., p. 478; Last Chance Co. vs. Tyler Co., 157 U. S., p. 683.

⁵ End-Lines and Side-Lines in the U. S. Mining Law, *Trans.*, xvii, 787 (1888-89).

manifold that it is not only possible but even probable that a second attempt might have succeeded. However, if such second attempt did fail there would be the rights of the Sims Fraction and Overlap to be taken into consideration. To the layman it might seem absurd to assume that either of these claims could possess rights to an orebody more than a half mile away and 1,700 ft. below the surface; but it is owing to a decision of the Department of the Interior that such claims were located.

The papers in this case are not accessible to me, but the history of it is this: Immediately west of the Viola and San Carlos was the Oakland, which overlapped both the former claims. The Viola was patented as shown; but when the San Carlos was patented the conflicting area of the Oakland was excluded. Later, when patent to the Oakland was applied for, the Department decided that the Oakland could not claim more than the total length of "free apex," and the east end line of this claim had to be drawn in, with the result that two fractions were left, which were located and patented as the Overlap and the Sims Fraction.

The Cheyenne would have given rise to a case very similar to the Drumlummon.

The remaining claim, the Lincoln, was located to acquire an open right immediately west of the King Fraction. This location, like the Copper Trust of Butte, was made on some widely separated fractions. But, while the claims of the Copper Trust were denied by the Supreme Court of Montana, that decision in no wise settled the rights claimed by the Lincoln, and the following opinion written by two eminent Colorado lawyers seems pertinent. They say:

"It has been said that hard cases make bad law. This seems to be particularly applicable to the Copper Trust case. The only surface open to location, and therefore appropriated by the Copper Trust location, was two exceedingly small triangles at the extreme easterly end of the location, only one of which, and that only for a slight distance, contained any part of the apex of the vein. The orebodies claimed by the Copper Trust under its extra-lateral right lay to the southwest of the Smokestack claim, and therefore within the extra-lateral right claimed by reference to the apex of the vein lying in the extreme westerly portion of the Copper Trust location. The

Dates of Location of Mining Rights on the Wardner Vein (see Fig. 2).

Bunker Hill.....	Sept. 10, 1885.	Republican Fraction..	Nov. 1, 1885.
Phil Sheridan.....	Sept. 16, 1885.	Amended.....	May 11, 1886.
Last Chance.....	Sept. 17, 1885.	Viola.....	Feb. 20, 1886.
Stemwinder.....	Sept. 17, 1885.	San Carlos.....	Apr. 23, 1886.
Amended.....	May 25, 1887.	Amended.....	May 11, 1886.
Emma.....	Sept. 17, 1885.	Cheyenne.....	Oct. 25, 1891.
Lackawana.....	Sept. 18, 1885.	King Fraction.....	June 22, 1898.
Tyler.....	Sept. 20, 1885.	Sims.....	Jan. 8, 1901.
Sullivan.....	Oct. 2, 1885.	Overlap.....	Apr. 9, 1901.
Lincoln.....	Sept. 8, 1905.		

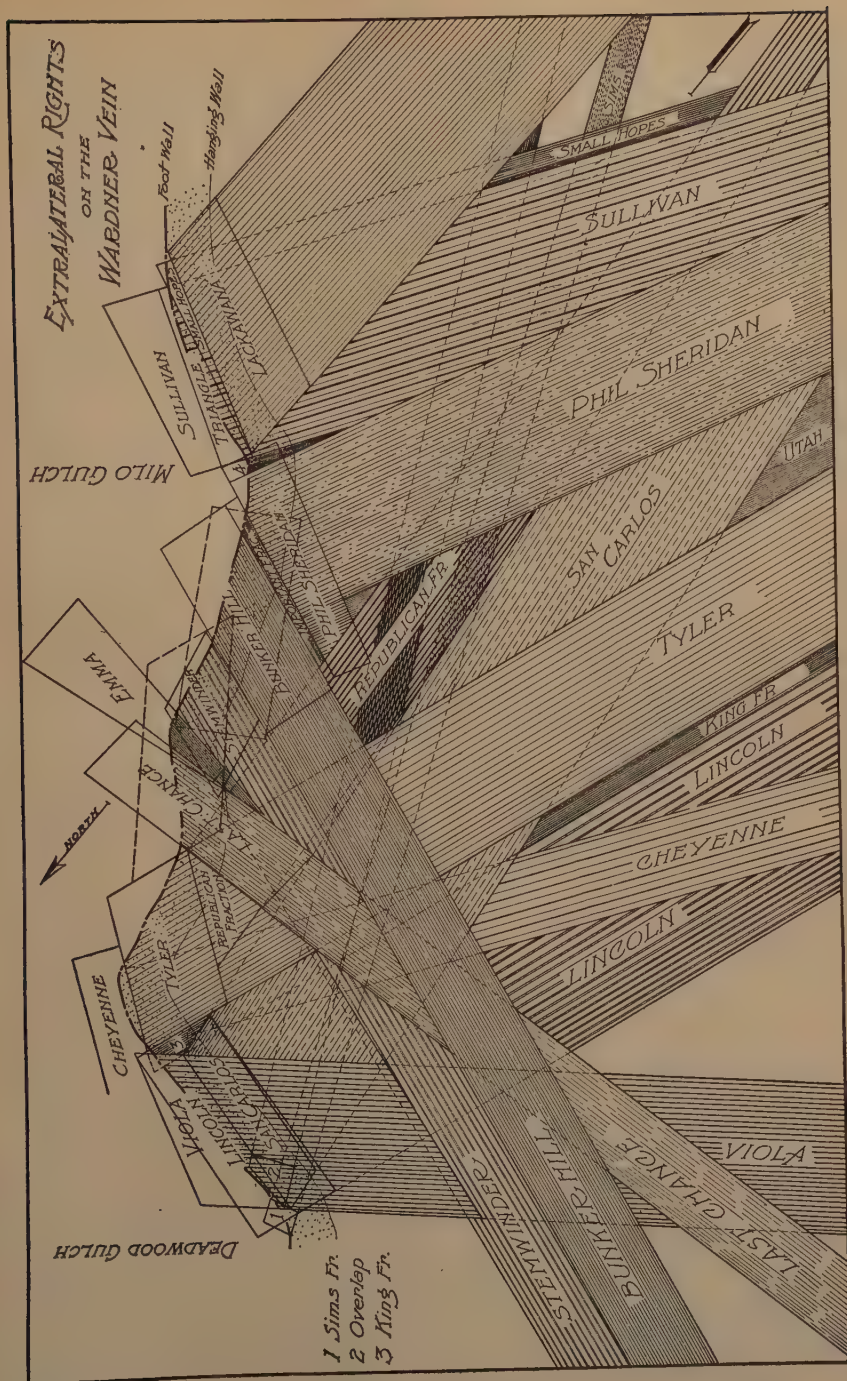


FIG. 2.—EXTRA-LATERAL RIGHTS ON THE WARNER VEIN. SCALE, 1 IN. = 1,
(See tabulation of dates of location on opposite page.)

matter complained of was an unusually broad order for inspection and survey. The logic of the opinion is not persuasive. Its analysis of the Del Monte decision, as it seems to us, is inaccurate and incomplete. It cites as an authority (65 Pac., 1025) Sec. 780 of the first edition of Lindley on Mines, a book issued prior to the decision in the Del Monte case and expressing views subsequently modified by the author in view of the Supreme Court's later decision.

"It is asserted in the Copper Trust case that in any event the Del Monte case is not applicable when the senior claim has gone to patent before the location of the

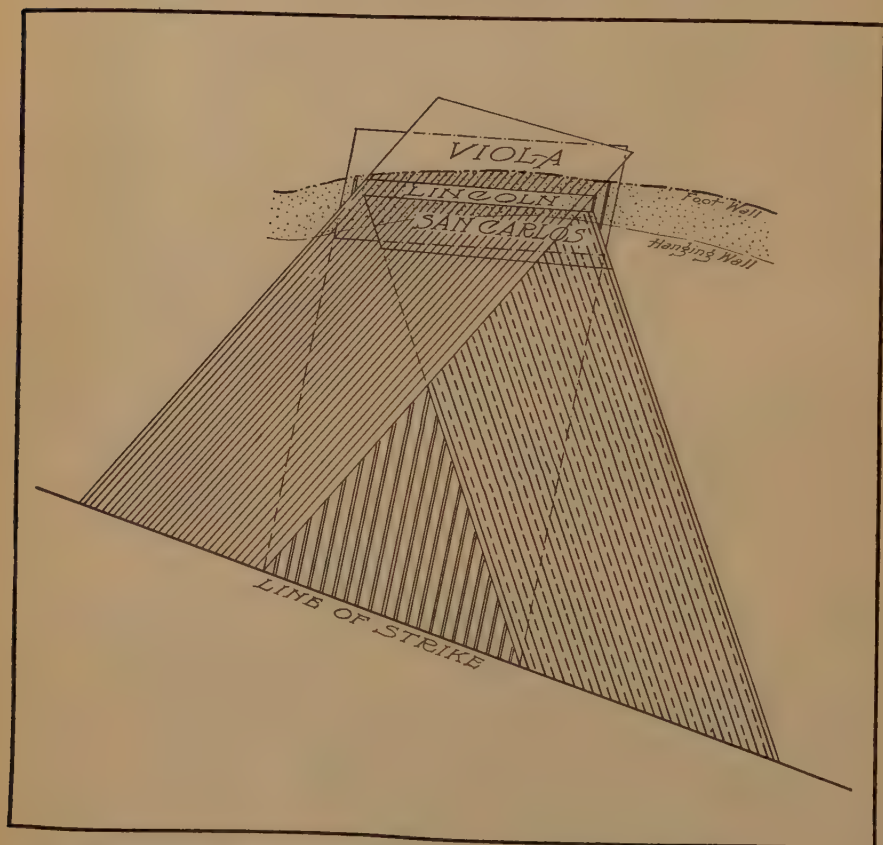


FIG. 3.—CONDITIONS RESULTING FROM A 'HYPOTHETICAL SEPARATION OF THE SAN CARLOS AND VIOLA MINING RIGHTS. SCALE, 1 IN. = 1,000 FT.

junior claim, but this is certainly in conflict with the authorities and erroneous on principle. The land department takes the opposite view (*The Hidee Gold Mining Co.*, 30 L. D., 420). The contrary view is certainly expressed by the Circuit Court of Appeals in the *Stemwinder* cases, and Judge Lindley says: (*I Lindley on Mines*, 2d ed., Sec. 363a, p. 657.)

"There is no reason which could be urged in support of permitting junior locators to lay the lines of their claims across prior unpatented claims which could not be invoked in behalf of a similar doctrine as applied to patented claims of all classes."

Whether this opinion be correct or not, the fact that it is the expression of reputable attorneys is a guarantee that it would be entertained by a court of law; and a long and expensive law suit would result.

It is easily conceivable that the San Carlos and Viola might have been separated by ten or more feet, and the condition shown in Fig. 3 would have existed. In this event there is little doubt but that the Lincoln would have had extra-lateral rights as shown and thus a claim of 1,500 ft. along the apex of the Wardner lode would have controlled rights to 4,100 ft. along the strike, at a depth of about 1,500 ft. (One of the "absurdities," no doubt, which Dr. Raymond had in mind.)

In the earliest of the many valuable papers contributed by Dr. Raymond to the subject of American mining law,⁶ is a map (p. 436) which bears a remarkable resemblance to Fig. 2 herewith given; and this coincidence is the more remarkable, since Dr. Raymond's paper was presented at the Troy, N. Y., meeting in October, 1883, more than two years before the first discovery was made in the Wardner district.

The litigation at Wardner affords not only a convincing example of the inadequacy of "the law of the apex," but clearly demonstrates that this method of acquiring the right to a mining location, based on a legitimate discovery, frequently fails to secure to the discoverer the vein he sought to locate.

The operators who have had any experience with suits arising from the law of extra-lateral right agree that the cost of such litigation is an irksome burden, and often, rather than incur the costs of such a suit, or venture into a court of law which is at variance with itself, they compromise with their neighbors, even though convinced that they are not getting all to which they believe themselves entitled.

On the other hand the lawyer has to rely upon the geologist for his facts as to structure, etc. It is conceivable that a mining geologist may become obsessed with a theory which he will apply to certain conditions, entirely neglecting some important fact which might destroy the logic and admissibility of his theory. The idealist will claim that any mistake of this kind would be eliminated before a case depending upon such a theory could come to trial; but even if this were so, there has been a great injustice done when it is possible to bring suit upon such evidence, and enjoin the defendant company from working parts of its mine, if it develops later, when the suit is dismissed, that the defendant company can get no commensurable recompense for the damage done it.

Naturally, the law is not exclusively responsible for these opportunities of the unscrupulous to use it for irreparable injury to others; but the opportunities will always exist, and the mischievous and preposterous predicament will be possible, so long as we are bound by the law of extra-lateral right. Dr. Raymond sums up the question very pertinently:

⁶ The Law of the Apex, *Trans.*, xii, 367 (1883-84).

"If all mining properties presented this beautiful simplicity of structure (ideal locations on ideal veins), and all mining locators exhibited a corresponding simplicity of purpose, the application of the law would be easy, but the naïveté of the statute fares badly between the freaks of nature and the tricks of man."

With the development of the science of ore deposits, our belief in the "freaks of nature" is rapidly fading, and many of the phenomena styled "freaks" are explainable. However, the other horn of the dilemma, while as easily explained as these alleged freaks, is just as hard to contend with as formerly, and under the existing mining laws there is small chance of circumventing these "tricks of man."

The law works badly in another way. Its language is decidedly untechnical; and ever since the first apex suit the courts and witnesses have tried to provide technical definitions for many of its terms and have succeeded only in clouding the meaning intended by the framers of the law. Were Milton a member of the Committee on Mining Law he would probably apply his own language to the law of the apex:

"Chaos umpire sits,
And by decision more embroils the fray
By which he reigns; next him high arbiter,
Chance governs all."

The Occurrence of Covellite at Butte, Mont.

BY A. PERRY THOMPSON, A. M., BUTTE, MONT.

(San Francisco Meeting, September, 1915)

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I. GENERAL DISTRIBUTION, APPEARANCE, AND RELATIONS

MINING in Butte has seldom encountered covellite in commercial quantities. The notable occurrences, extending vertically and laterally perhaps several hundred feet, presented scattered, bladed lenses of this mineral in a matrix of quartz, pyrite, enargite, bornite, and chalcocite. But under the microscope covellite is seen to be widespread in the Butte ores, though in such small quantity that megascopic examination will not discover it.

In early mining operations covellite was found in the Gray Rock vein, a member of the Blue vein system. As described by Sales,¹ it persisted as an occasional ore mineral, associated with quartz, pyrite, tetrahedrite, bornite, chalcocite, and minor amounts of sphalerite, from the 700-ft. to the 1,200-ft. level of the East Gray Rock and Diamond mines. Crushed quartz-monzonite and later quartz completed the vein filling between

¹ Reno H. Sales: Ore Deposits at Butte, Mont., *Trans.*, xlvii, 3 to 109 (1913).

strong clay walls, sometimes as much as 20 ft. apart, while in other places along the strike the vein pinched to a few inches of clay. The mineralization was a replacement of crushed quartz-monzonite between the walls, and formed an ore shoot several hundred feet long, pitching to the SE. and terminating upward, 500 ft. below the surface.

Later, intermittent bunches of covellite were encountered in the Speculator mine, on the Edith May vein, also a member of the Blue vein system. In the vein matter which was capped by barren fault clay and crushed quartz-monzonite, the covellite continued from the 700-ft. to the 1,600-ft. level and laterally several hundred feet. Associated with it in the fault vein were the minerals shown in the Gray Rock vein, with the addition of considerable enargite. Tetrahedrite was found in sparse but universal distribution. The old stopes, in these ore bodies of the Gray Rock and Speculator mines, have long been abandoned and no record has been preserved of their structural or mineralogical details.

Minor amounts of covellite have been found in the veins of the Mountain View, Tramway, Diamond, High Ore, Bell, and Buffalo mines. It appears where conditions in the primary mineralization were favorable for the development of small amounts of the mineral, frequently in crystal form and in vugs, associated with pyrite, enargite, sphalerite, tetrahedrite, bornite, chalcopyrite, and chalcocite. The association of minerals is very similar to that observed with the microscope in the massive ores when the covellite is very small.

In the sooty chalcocite, the presence of covellite is often attested by an indigo-blue tarnish and a bluish powder, frequently present in the enriched portions of all the copper veins. This type of covellite marks an intermediate stage in the enrichment of the primary copper ores and bears no relation to the bodies of primary cupric sulphide below.

In the Leonard mine, large stopes are now worked on the Colusa-Leonard vein, a member of the Anaconda system.² In these stopes, for 600 ft. below the 1,400-ft. level, covellite can occasionally be seen in an area of quartz-monzonite which has been subjected to intense replacement. Here the upper limit of covellite occurs near the 1,000-ft. level, and shreds and stringers of bladed masses in the vein filling have been frequently found along the strike of the orebody.

This unusual development of covellite lies in an area of intense alteration and rich mineralization of the quartz-monzonite. In the portion of the Leonard vein now exposed, where covellite occurs, the "horse-tail" structure of the orebody is well defined. The course of the main mineralization and fissuring is in general E-W.; but along the vein where covellite has been noted, many seams and veinlets of ore strike off to the south. The whole mass of the country rock, a normal quartz-monzonite,

² R. H. Sales: *loc. cit.*, p. 14.

has been thoroughly shattered south of the north wall of the Colusa-Leonard vein. The main zone of fissuring has been developed along the E-W. orebody. Most commonly the main fissures and minor horse-tail seams, spraying off to the south, are within a few degrees of the vertical, since the strongest movement has taken place in that direction, but the resultant forces produced many minor cracks in every conceivable direction between the larger fracture planes. These cracks are all very irregular and frequently of almost microscopic width. The system of fractures has formed a passageway for mineralizers coming from below and from the main fissure zone and is filled with mixtures and pure veinlets of quartz, pyrite, enargite, covellite, and chalcocite. The quartz-monzonite, generally speaking, is very solid. Veinlets show no displacement or crushing effects. Vugs are often seen in the veinlets and adjacent rock. In the main E-W. fracture zone and between the horse-tail seams and cracks, the quartz-monzonite is sericitized, pyritized, and silicified. Areas of country rock in the zone of intense fracturing have been thoroughly permeated by the early mineralizers. At high temperatures the quartz-monzonite seems to have been comparatively permeable to the solutions causing sericitization, pyritization, and silicification. That the mineralizers continued to obtain access to the solid country rock through the period of ore deposition is evidenced by the abundant replacement of the ferro-magnesian minerals by enargite and by lenses of chalcocite along minute fractures in the solid rock.

Enargite is the most abundant copper mineral in the covellite-bearing areas. It is found replacing the quartz-monzonite in solid veinlets from microscopic size to 3 or 4 ft. in thickness. These larger masses, when broken open, show the enargite well crystallized throughout the entire veinlet. Often smaller stringers in the solid rock have a fine lining of quartz on each side of the enargite filling. Such quartz linings are at times without definite crystal development and again exhibit perfect crystalline forms extending into the enargite, affording evidence that the whole mineralization took place in previously formed cavities. Many enargite veinlets do not have a quartz lining and afford fine examples of replacement of quartz-monzonite along fracture planes. Veinlets several inches thick show no pyrite. Some (always small) cracks are filled with pyrite alone. From the undisturbed appearance of these cracks and replacement seams, it is inferred that there was little movement in parts of the horse-tail area after the original shattering of the country.

Covellite blades, often several inches long, occur in the enargite veinlets, which often show no crushing such as might have afforded an opening for the later deposition of covellite. The cupric sulphide may be present in one foot of a veinlet, while the next foot, in either direction, may pinch out or be pure enargite. In many instances, there is no apparent connection between the masses of covellite. Covellite also occurs in veinlets in



FIG. 1.—COVELLITE STRINGERS THROUGH ENARGITE, PYRITE, AND QUARTZ THAT HAS REPLACED QUARTZ-MONZONITE. V-SHAPED HORSE OF COUNTRY ROCK AT BOTTOM OF VIEW. NATURAL SIZE.

Co, covellite; E, enargite; G, gangue; P, pyrite.
1,600-ft. level, Leonard mine, Leonard vein.

quartz-monzonite, without apparent association with enargite, and in bladed crystals replacing the country rock, sometimes several inches from veinlets of any sort.

In Fig. 1, a stringer of covellite blades may be seen running through enargite, quartz, and pyrite that have replaced the quartz-monzonite. A V-shaped horse of altered country rock may also be seen. The ferromagnesian minerals have been replaced by enargite, chalcocite, and covellite. The quartz and feldspar are still partly preserved. Fine veinlets of enargite, covellite, and chalcocite ramify through the included rock. Residual quartz crystals still remain in the covellite. The cupric sulphide has replaced the quartz-monzonite, enargite, and pyrite.

In the portion of the quartz-monzonite showing the most crushing and alteration, there is no covellite. Here the development of quartz, pyrite, enargite, and chalcocite is greatest. Large amounts of chalcocite replace the previously formed minerals. Near the zone of greatest movement are found many more vugs and a greater development of silica, together with a more marked alteration of the wall rock.

It appears that after the original fracturing, solutions penetrating the crushed zones dissolved parts of the quartz-monzonite and to a considerable extent formed solution channels and vugs. The alteration of the country rock was most intense along the E-W. lines of greatest fracturing and in them the greatest silicification occurred. In this zone of most intense fracturing, in the vugs and in the fractures adjacent to the main crushed zone, were deposited first, quartz and pyrite, and probably contemporaneously or nearly so, a large amount of enargite. South of this zone, in the horse-tail area, the quartz-monzonite has been greatly shattered and the cracks have been chiefly filled with, and the country rock replaced by, pyrite and enargite. Quartz is frequently developed as a lining to the enargite veinlets and pyrite occurs sparingly in seams.

From the appearance of these veinlets and vugs, we infer that there was no interval of time, or cessation of mineralization, during the deposition of quartz, pyrite, and enargite. It appears most likely that the mineralizing solutions depositing these minerals entered the shattered quartz-monzonite after its alteration by solutions which caused sericitization and dissolution and deposited their burden in the zone of main fracturing. Quartz and enargite predominated in the early mineralization. Considerable enargite continued to be deposited after the quartz had solidified and had been crushed. Solutions permeated the shattered horse-tail area, depositing enargite after the replacement of the quartz-monzonite with earliest minerals. At this time many of the veinlets were formed in the minor cracks, some of which were due to solution, and in such fashion that a lining of silica was precipitated before the main filling with enargite.

Many stringers of enargite, up to 2 ft. in width, show no chalcocite.

These bodies of enargite frequently contain inclusions of quartz-monzonite and residual quartz, affording conclusive evidence that the stringers involve replacements of the country rock and are not merely filled fissures.

Final solutions ascending from the source of mineralization were very active in dissolving enargite and covellite. Many vugs are found where these two minerals have been attacked by later waters. Enargite particularly was susceptible to the action of the later solvents. Skeletons of enargite crystals are commonly preserved. These vugs universally contain plentiful but small crystals of barite. It is believed that the solvents attacking the enargite and covellite were comparatively alkaline and that the presence of barite, later than the covellite and possibly chalcocite, adds strong evidence to their determination as primary minerals.

The above discussion deals only with that part of the Leonard vein where covellite is found. Other areas of horse-tail ore exhibit a different association of minerals. Much of the enargite of the horse-tail region has been attacked by mineralizers which were depositing chalcocite. Stringers of glance from the finest dimensions up to 2 ft. in width are found, which were precipitated by a direct replacement of solid quartz-monzonite. At times the country rock is peppered with shots of chalcocite, similar to the distribution of enargite described above.

II. DATA FROM MICROSCOPIC EXAMINATION

1. *Criteria for Distinguishing Primary and Secondary Minerals.*

The question of the primary or secondary character of the deep chalcocite orebodies in Butte has long been a matter of argument. Sales³ was the first to furnish conclusive proof, both structural and petrographic, of the primary deposition of chalcocite in all periods of the Butte vein formation. Several investigators, through the microscopic examination of polished sections of the copper ores, have discovered features peculiar to each type of chalcocite. The results promise to make possible the microscopic determination of its primary or secondary nature.

In the study of the bornite and chalcocite ores of the Virgilina district of North Carolina and Virginia, Laney⁴ found graphic intergrowths of bornite and chalcocite resembling the pattern of an eutectic alloy. Similar structures were found in the bornite and chalcocite of the Guilford mine, Guilford County, North Carolina, by Graton and Murdoch.⁵ Eutectic

³ *Loc cit.*, p. 93.

⁴ F. B. Laney: Relation of Bornite and Chalcocite in the Copper Ores of the Virgilina District of North Carolina and Virginia, *Economic Geology*, vol. vi, No. 4, p. 407 (June, 1911).

⁵ L. C. Graton and Joseph Murdoch: The Sulphide Ores of Copper: Some Results of Microscopic Study, *Trans.*, xlv, 76 (1913).

structures in the development of bornite and chalcocite intergrowths have been found in the ore of the Leonard, Jessie High ore and Anaconda veins at Butte.

Bornite in the Leonard vein is associated in small quantities with covellite. Instances of the occurrence of eutectic patterns of bornite and chalcocite were observed where the alteration of bornite to chalcocite had not progressed far. The eutectic pattern, though seldom observed, gives strong evidence for the primary nature of the chalcocite. The Jessie vein, described by Sales, belonging to the Blue fault system, is unusual in its mineral composition. The oreshoot furnishing the material examined shows a complete transition, between the 500-ft. and 800-ft. levels, from an ore consisting of quartz, pyrite, and chalcopyrite to a mixture of quartz, pyrite, enargite, chalcopyrite, and chalcocite. In ore from this lower zone of mineralization, graphic intergrowths of bornite and chalcocite were found. Bornite, enargite, and chalcopyrite, all undoubted primary minerals, contained irregular patches of chalcocite, to all appearances primary, but later than the bornite. This occurrence merges into prominent developments of eutectic structures.

An eutectic alloy consists of an intimate mechanical mixture of two solid phases, produced by crystallization at a constant temperature. The parallel arrangement of the two phases indicates that the crystallization is not simultaneous, but that the molecules of the two phases crystallize alternately. This deposition begins at a number of centers and the interference of the crystal systems leads to the formation of polyhedral masses, composed of an intergrowth of the two phases. The crystallization of bornite and chalcocite in this system of eutectic separation presents strong evidence of the simultaneous deposition of the two minerals and of the primary character of the chalcocite.

Primary intergrowths of bornite and chalcocite have been described by Laney,⁶ and by Graton and Murdoch.⁷

Such an intergrowth is a characteristic mode of occurrence of many of the primary copper minerals of Butte. Enargite and bornite commonly exhibit this structure. Bornite and chalcopyrite are found similarly intergrown. Primary intergrowth presents an irregular interlocking of different minerals, generally without strongly developed crystallization—a meshy, entwined mass of one mineral intergrown with another, preserving at the same time, sharp mineralogical contacts. This primary intergrowth structure is commonly found in eutectic alloys. Eutectics of copper and copper phosphide, or of bismuth and lead, often develop structures identical to that found in the primary intergrowths of the copper ores. As a proof of primary origin of chalcocite, this type is no less conclusive than the more perfect eutectic with parallel arrangement.

⁶ *Economic Geology*, vol. vi, No. 4, p. 406 (June, 1911).

⁷ *Trans.*, xlv, 77 (1913).

Graton and Murdoch⁸ found that chalcocite, believed from all other evidence to be primary, yielded, when etched, a definite type of structure. The primary, deep-level chalcocite of Butte, in a modified form, exhibits etchings similar to that mentioned above. The general characteristics of this etching include a symmetrical development of coarse-grained cleavage. Often, when acid is applied, perfect regularity in faces and angles is lacking, and for general practice, a structure involving straight, sharply defined boundaries and regular orientations for each individual grain, together with a systematic arrangement of columnar and other grains, is evidence of the primary character of the glance. The surface of such a specimen will present a smooth, mosaic effect after etching, with a definite and symmetrical outline for each grain. (See Fig. 2.)



FIG. 2.—PRIMARY CHALCOCITE ETCHING STRUCTURE. $\times 80$.

C, chalcocite; P, pyrite.

1,600-ft. level, Leonard mine, Leonard vein.

Chalcocite from the 200-ft. level in the Copper Queen mine from the Sacramento Hill porphyry orebody exhibited a splendid example of this primary chalcocite etched structure. Orthorhombic cleavage developed in the etched chalcocite a far different structure from that of the secondary chalcocite of other orebodies.

An entirely different type of structure is evolved when secondary chalcocite is etched. This appearance is believed to be due to the molecular arrangement of the chalcocite and its manner of formation. At least two

⁸ *Trans.*, xlv, 79 (1913).

different types of replacement may be distinguished in the formation of secondary glance. The first type includes characteristic veinlets of secondary chalcocite ramifying through the primary ore. These veinlets may vary from minute cracks to masses of such size that the original ore entirely disappears. In the development of the cracks certain features are prominent. Almost invariably the fracture from which enrichment progresses is preserved in the center of the veinlet, either as a fracture or as a tabular body of copper minerals or of gangue. Most commonly, the alteration of the primary ore involves the production of intermediate products which form a transition zone between the original ore and the final development, in the veinlet, of chalcocite. This vague cloudy border and the dark medial line are typical of secondary chalcocite. (See Fig. 3.)

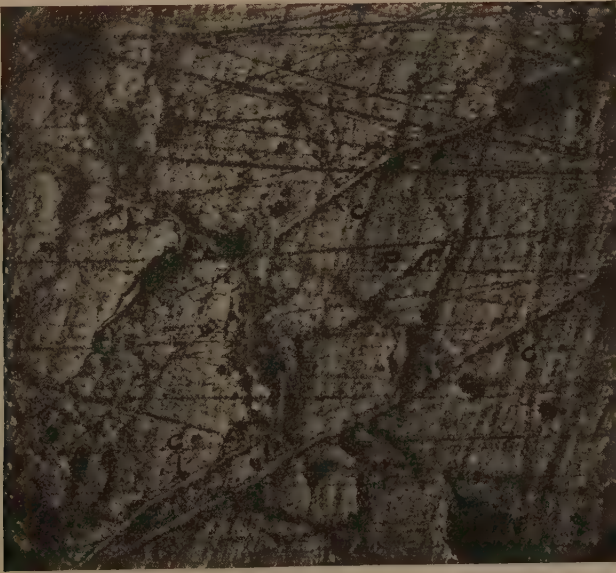


FIG. 3.—PYRITE SEAM IN WALL ROCK. SECONDARY CHALCOCITE STRINGERS IN PYRITE. $\times 80$.

C, chalcocite; P, pyrite.
300-ft. level, Silver Bow mine.

A second type of secondary enrichment is found in soft, friable masses of pyrite. Frequently the action of descending sulphuric acid or sulphate solutions tends to render solid pyrite very friable and porous. When deposition of secondary chalcocite takes place in such veins, the action does not proceed from fractures but from many points throughout the entire mass of pyrite. These conditions tend to build up separate grains of sooty glance around many centers, each individual developing a botryoidal mass. Continued action results in lenses of massive glance, more brittle and of much darker luster, generally, than primary glance, with a notable

development of botryoidal, stalagmitic structure (Fig. 4). Lenses of such material frequently contain columnar and stalagmitic forms 6 in. long, the



FIG. 4.—BOTRYOIDAL SECONDARY CHALCOCITE WITH REMNANTS OF ORIGINAL PYRITE VEIN FILLING. FULL SIZE.

C, chalcocite; P, pyrite.

400-ft. level, Berkeley mine, Snohomish vein.

whole molded together to make a compact ore. In the vicinity of the lenses, all stages of development of the structure may be observed, from the early sooty deposit on pyrite grains to solid botryoidal stalagmites.



FIG. 5.—BORNITE HALO AROUND PYRITE REMNANTS IN SECONDARY CHALCOCITE. $\times 80$.
 B, bornite; C, chalcocite; Ca, chalcopyrite; P, pyrite.
 400-ft. level, Berkeley mine, Snohomish vein.



FIG. 6.—SECONDARY CHALCOCITE ETCHING STRUCTURE. $\times 80$.
 C, chalcocite; G, gangue; P, pyrite.
 400-ft. level, Berkeley mine, Snohomish vein.

In the enrichment of pyrite by descending acid, cupriferous solutions, chalcopyrite is first developed as a halo around the pyrite. Bornite next forms, around the chalcopyrite; with the further addition of copper, the final stage is chalcocite. The series furnishes a record of the orderly addition of copper and of the subtraction of iron. Remnants of pyrite surrounded by halos of secondary chalcopyrite and bornite may be seen in Fig. 5.

The above structures have been found to afford etch figures entirely different from those of primary chalcocite. When secondary chalcocite is etched, lines are developed in very ragged fashion. Etching into regular grains is seldom seen. The whole mass presents a disintegrated granular mineral matrix. (See Fig. 6.) Corroborative evidence of the soundness of this structure as a test for secondary chalcocite has been obtained by etching secondary chalcocite from the Briggs, Shattuck, and Junction mines of Bisbee, Ariz. A notable feature of the etching of massive secondary chalcocite is the scaly overlapping of sections between cracks (Fig. 6). This may be called a fish-scale structure. It is due to deeper etching on certain sides of the grains than on others. Such variation in the attack of the acids is believed to be due to the botryoidal texture.*

2. *The Leonard Vein Filling*

The copper minerals of the Leonard vein bear a constant relation to one another in form and paragenesis. Examination of many polished sections from the horse-tail ore leads to conclusions concerning the formation of the ore minerals that appear generally applicable.

Evidence in the hand specimens points to the deposition of quartz, pyrite, and enargite, as the first stage of mineralization. Quartz was formed first and continued to be deposited after the crystallization of enargite. Similarly, pyrite is found of a later age than enargite. In places, principally along the main E-W. fracture zone, considerable quartz and pyrite were formed before slight shattering admitted cementing solutions of enargite. Examples are plentiful of the replacement of the ferro-magnesian minerals first, and later of all the rock-forming minerals of the quartz-monzonite, by quartz, pyrite, and enargite. Enargite has been particularly active in this respect, filling solution channels, vugs, and fractures.

* Since the above determinations on etching structures of primary and secondary chalcocite were made in April, 1914, new criteria have been developed. Perfect orthorhombic etch figures have been obtained from undoubted secondary chalcocite, from the 300-ft. level of the Silver Bow mine. Other similar instances combine to place doubt upon the infallibility of the etch test for the primary or secondary nature of chalcocite. It is a well-established fact, however, that the deep-level chalcocite of all the Butte veins produces similar distinctive etch figures. Often the curved mosaic pattern is due to cleavage inherited from replaced bornite.

Quartz.—In polished sections, quartz is seen to have been deposited first, and also, in minor amount, during the successive stages of mineralization. At times good hexagonal crystals, bounded by the copper minerals, are preserved; and again, irregular grains show signs of alteration by the ascending solutions. Quartz, with pyrite, appears to be the material most resistant to alteration. Some quartz remnants in the vein are original crystals from the quartz-monzonite.

Pyrite.—Pyrite in small and slightly shattered grains is peppered through the ore. It is surrounded and replaced by enargite, covellite, chalcocite, bornite, and chalcopyrite. These remnants of former pyrite crystals have been strongly attacked by later solutions. Alteration is



FIG. 7.—INITIAL ALTERATION OF COVELLITE BY CHALCOCITE. THE MINUTE SPECKS ARE SURVIVING GANGUE. $\times 80$.

C, chalcocite; Co, covellite; P, pyrite.
1,600-ft. level, Leonard mine, Leonard vein.

evidenced by the corroded rough outlines of the former crystals and by areas of the later copper minerals inclosed in crystals of the iron sulphide. Solutions which deposited covellite and chalcocite have been particularly active in attacking the pyrite. Results of this dissolution show rounded remnants of pyrite grains imbedded in covellite and chalcocite. (See Fig. 7.) Solutions containing enargite have also attacked the pyrite. When enargite, covellite, or chalcocite is found within crystals of pyrite, frequently no feeding fracture for this mineralization can be observed. It is believed that the mineralizers have gained access to the interior of

pyrite grains along lines of weakness or cleavage and have replaced the iron sulphide directly, volume for volume. Many small masses of chalcocite thus formed in large crystals of pyrite, show no indication of bornite or chalcopyrite as an intermediate stage in the alteration.

Enargite.—By reason of its early deposition and abundance, enargite shows the effect of all the later changes in the veins. Little movement is evidenced by the enargite veinlets. It is probable that later movement was largely along the E-W. fractures, leaving the enargite stringers of the horse-tail ore, in general, undisturbed. Many stringers of enargite in the horse-tail ore show absolutely no signs of movement since formation, yet, within these veins, lenses and bunches of covellite blades are inter-

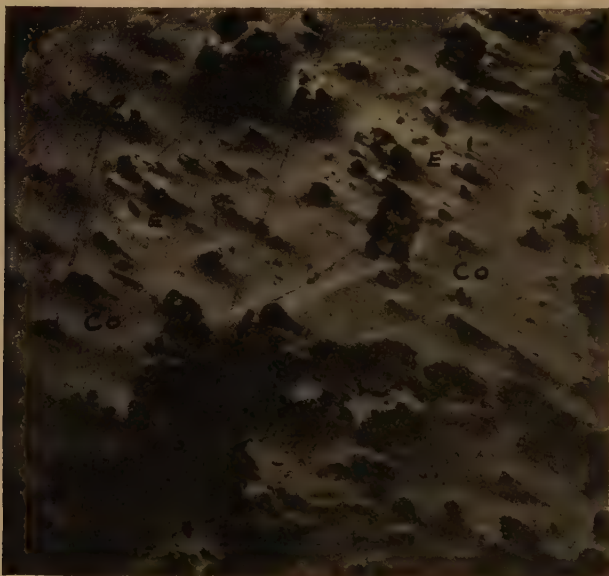


FIG. 8.—ENARGITE AND QUARTZ REMNANTS IN COVELLITE. THE MINUTE SPECKS ARE SURVIVING GANGUE. $\times 80$.

Co, covellite; E, enargite; G, gangue.
1,600-ft. level, Leonard mine, Leonard vein.

mittently distributed. These stringers are the result of direct replacement of quartz-monzonite, and of the filling of vugs, solution channels, and fractures. Along minute joint planes in the country rock, the ferromagnesian minerals are found replaced by enargite, the feldspars and quartz unattacked. Processes of replacement continued, finally attacking the feldspars and quartz to form the larger veins, sometimes several feet wide. Persistent specks of gangue, occurring in considerable quantity, in the ore, are remnants of the more resistant quartz-monzonite minerals. Quartz was deposited during this period of mineralization

and is present as included matter in the veins and as a fine lining to the veinlets.

The enargite has been much corroded and in places entirely replaced by later solutions depositing covellite. (See Fig. 8.) Remnants of enargite of varying size, surrounded by covellite and glance, are frequent. (See Fig. 9.) Covellite stringers through enargite further attest the alteration by later solutions. Often the crystal faces of the sulpharsenide have acted as a barrier to the invasion of covellite and form clear-cut boundaries for the molding of the later minerals.

Minute fracture planes, cleavage faces, and cavities between crystals, have united to produce a porous structure in some of the deposits of

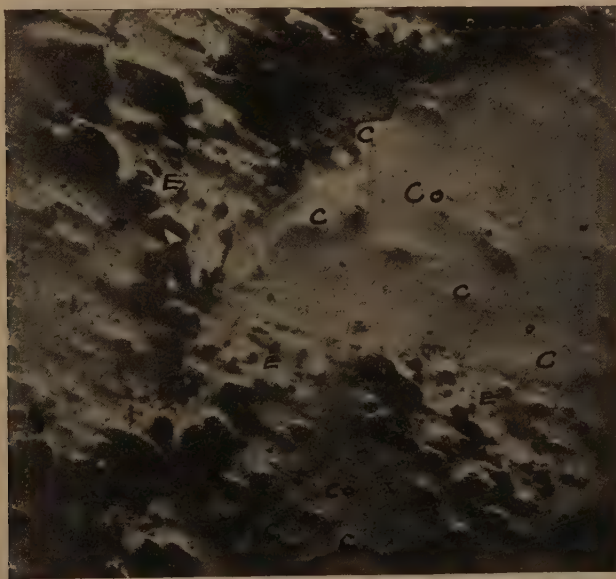


FIG. 9.—REPLACEMENT OF ENARGITE BY COVELLITE AND CHALCOCITE. $\times 80$.

C, chalcocite; Co, covellite; E, enargite.
1,600-ft. level, Leonard mine, Leonard vein.

enargite. Moreover, in the Leonard mine, the deposition of covellite appears to have been limited to enargite areas. Such a selective action is believed to be due chiefly to structural features and local conditions of the mineralizers. In a cross-section of the Leonard vein, bunches of covellite will exhibit a decidedly local arrangement. Much enargite, containing no covellite, has been mined. Country rock in the vicinity of covellite sometimes contains small crystals of the cupric sulphide. Perfectly formed covellite crystals occur in vugs of the veins and of the quartzmonzonite.

The part that enargite has played, from physical evidence, may thus

be said to be that of a favorable medium for replacement and the deposition of other copper minerals, governed by the porosity of the veins and the local character of the mineralizers.

Chalcocite has been very active in the replacement of enargite. With this phenomenon a direct transition from the arsenical sulphide to glance occurs, as illustrated by Fig. 10. Sometimes the transition zone is marked by a bright white rim surrounding the enargite. Again, small areas of enargite in the process of alteration to chalcocite present this appearance. The white material probably represents an intermediate



FIG. 10.—REPLACEMENT OF ENARGITE BY CHALCOCITE. $\times 80$.

C, chalcocite; E, enargite.

1,600-ft. level, Leonard mine, Leonard vein.

stage of alteration, either chemical or microscopically physical, between the sulph-arsenate and the cuprous sulphide. With the structure of enargite it displays the color of chalcocite and a peculiar brightness of its own. No such zone is observed in the alteration of enargite to covellite.

Coincident with the replacement of the quartz-monzonite minerals, more or less perfect crystal form has been preserved in enargite. Previously deposited quartz and pyrite have influenced enargite in its crystallization; but the orthorhombic angles and faces adjust themselves to this limitation.

Covellite.—By far the larger part of the covellite in the Leonard ore-body is a direct replacement of enargite by ascending primary solutions—

the upward secondary sulphide enrichment of Tolman⁹ and Rogers.¹⁰ As already shown, the deposition of covellite has been governed by structural, physical, and chemical conditions. The most pronounced development in the Leonard vein has occurred in that part of the E-W. mineralization where the horse-tail stringers begin to spray off to the south. Locally, small lenses and stringers of covellite have been formed for 20 or 30 ft. south in the horse-tail veinlets and in the country rock. In this connection, it must be remembered that crystals and minor lenses of the cupric sulphide have been found in the horse-tail ore, hundreds of feet south from the north wall of the Leonard vein. The porosity of the enargite, slight fracturing of the veins in the country rock, and replacement of the quartz-monzonite and vein minerals, provided space for the crystallization of covellite.

Too much stress cannot be laid upon the penetrating power of mineralizers. When apparently solid country rock has been highly altered a hundred or more feet from the veins or fracture systems, we must conclude that a mineralizing solution or an igneo-aqueous vapor can, under the prevailing temperatures and pressures, penetrate porous veinlets of enargite and shattered areas. That local conditions in the mineralizers largely accounted for the formation of covellite, after the deposition of the enargite had ceased, may be seen in the fact that large amounts of enargite in the vein, above, below, and laterally, contain no covellite. Such conditions, together with structural and physical influences, have produced areas in the vein where other minerals have likewise been localized. Two hundred feet below this occurrence of covellite, persistent solid chalcocite veins, with only a slight sprinkling of pyrite, replace sericitized quartz-monzonite, at times, to a width of two or more feet.

Evidence of the replacement of enargite by solutions depositing covellite is found in the small and large remnants of enargite in covellite areas (see Fig. 8); in stringers of covellite through enargite crystals; and in irregular areas of covellite that have eaten into the interior of enargite masses. The action appears to have been mostly one of replacement, volume by volume. The enargite shows little evidence of fracturing.

Development almost wholly from enargite has left few characteristics of structure in the covellite. Even in apparently solid masses of crystal plates of covellite, unreplaced enargite may be seen with the naked eye upon close examination. Under the microscope, small and large portions of enargite are seen to be inclosed in the cupric sulphide. Covellite has at all times a very strong tendency to develop its own hexagonal or rhombohedral plates with perfect basal cleavage. These plates are found

⁹ C. F. Tolman, Jr.: Recent Advances in the Study of Sulphide Enrichment, *Mining and Scientific Press*, vol. cviii, No. 4, p. 172 (Jan. 24, 1914).

¹⁰ A. F. Rogers: Chalcocite Formation at Butte, Montana, *Economic Geology*, vol. viii, No. 8, p. 788 (Dec., 1913).

crossing the crystals of enargite in every direction and frequently grow to several inches in length, far exceeding the limits of enargite crystals, which seldom exhibit parallel orientation in adjacent individuals. That covellite has for the most part strongly preserved its own crystal form under conditions indicative of free crystallization, is everywhere apparent.

Shreds of the cupric sulphide are observable under the microscope and in large areas present peculiar phenomena. On etching with potassium cyanide, the structure of covellite is brought out, the cyanide salt eating into the cleavage cracks and leaving a rough dark network on the polished surface. The same salt attacks enargite in a similar manner. When areas of covellite, that have resulted from a replacement of enargite, are etched, a series of lines parallel to the basal cleavage is developed. Minor lines in all directions complicate an indefinite rough etching between major directions. When enargite grains are included in the etched area, the orientation of the crystal systems may be compared. Enargite cleavage stands out prominently when etched with potassium cyanide. Crystals of the sulph-arsenate are commonly found in covellite areas with orthorhombic cleavage lines oriented in all conceivable directions to the basal etching of covellite. Moreover, one blade of cupric sulphide may cut across half a dozen or more crystallographic units of enargite, preserving its own structure throughout. That this mineral is strongly addicted to crystallize with its own form and structure is plain from the similarity of the covellite that has replaced enargite, pyrite, and country rock, to the covellite crystals formed in the free space of vugs.

In this connection the universal mottled structure of covellite is of interest. Observed variations in color, which may be due to several physical causes, are evidenced by irregular intergrowths and penetrations of a light and a deep blue. At times the variation appears to be the effect upon previously formed dark-blue covellite of later light-blue material. Again, change in color may be due to different orientation in the grains of cupric sulphide. The presence of grains of enargite not completely altered to covellite, but appearing light blue, gives rise to a mottled effect and on etching presents the typical enargite crystal structure. Later shady developments of bornite produce a mottled appearance in sections. Much of the variation in color, however, is believed to be due to the chemical constitution of the covellite.

Sections of covellite blades typically show considerable finely divided gangue. (See Fig. 7.) To a large degree enargite presents the same phenomenon. (See Fig. 8.) The survival of the gangue in the covellite is probably due to the preservation, in the replacement of enargite by covellite, of the gangue particles already contained in the enargite. Some quartz may also have been deposited contemporaneously with the formation of covellite.

Subsequent to the deposition of the cupric sulphide, a deformation of

slight extent ensued, which in places bent, buckled, and fractured the blades, forming, along and across cleavage directions, angular spaces and lenticular gashes, which have been filled with later primary bornite and chalcocite. (See Fig. 11.) These later primary sulphides, including minor amounts of chalcopyrite, have greatly altered the covellite lenses.

Bornite.—Following the cessation of covellite deposition, and the later deformation of the vein minerals, bornite was precipitated from solutions, filling interstices in and between covellite, enargite, pyrite, and quartz, and at the same time replacing these minerals, particularly the covellite. The condition of bornite in all the Butte ores is very unstable. It evidently



FIG. 11.—SHATTERING OF COVELLITE BLADES AND CEMENTATION AND REPLACEMENT BY CHALCOCITE. $\times 80$.

C, chalcocite; Co, covellite.

1,600-ft. level, Leonard mine, Leonard vein.

replaced the enargite and covellite, at times to a considerable extent, but, being unstable in the presence of later chalcocite mineralizers, has almost disappeared. Its relation to chalcocite is very intimate and often difficult to unravel.

During the process of replacement of enargite and covellite, the gangue underwent little alteration. Its presence, in irregular narrow lines of bornite in chalcocite areas, has served to identify bornite as an earlier mineral. When all of the previous copper sulphides are replaced by chalcocite, the gangue disappears as well. In this respect the solutions depositing glance are believed to have been more energetic reagents than the earlier mineralizers.

Coincident with the replacement of covellite, a great deal of bornite developed as an intermediate stage in the alteration of older minerals to chalcocite. Bornite appears in shadowy, irregular areas and has, to an advanced degree, been replaced by glance. Often, after etching polished slabs of bornite and chalcocite which have originated by replacement, the two minerals are seen to grade into each other, with very delicate changes in color, and no definite demarkation between one mineral and the other. The copper-iron sulphide is found mostly as a rim around plates and remnants of covellite, as irregular areas at the contact of covellite and glance, and peppered through glance in very small grains, remnants which survived the processes of replacement. This later occurrence gives a mottled appearance and a rough relief to the chalcocite.

In the case of sections where bornite appears to have been formed definitely earlier than chalcocite, it is believed that when the ores were being deposited, the mineralizers were at first constituted so that bornite began precipitating out as the covellite was replaced, by reason probably of the abundance of iron. As this saturation for bornite lessened or the iron contents diminished, chalcocite was precipitated. That such a condition prevailed is evidenced by the earlier masses of bornite, the crystallographic intergrowths of the two minerals in the Leonard ore, and the universal primary intergrowths. Considering the conceded primary nature of bornite, these developments are strong evidence for the primary nature of the chalcocite and covellite. As the determining iron factor in the solutions lessened, replacement of covellite by chalcocite took place directly. It appears, however, probable that, even during the last stages of chalcocite deposition, bornite continued to be formed in small quantities.

In this period of alteration occurred a very minor development of primary chalcopyrite, which is believed to have been intermediate in the alteration of covellite and pyrite by chalcocite mineralizers, thus occupying a position like that of bornite, which it preceded or followed. Most of the chalcopyrite in the sections is formed near pyrite grains. These pyrite grains are believed to have furnished some iron for the formation of chalcopyrite and bornite by the ascending primary solutions.

As is evident in other Butte veins, much bornite is deposited directly from primary solutions, as an enrichment and as a cementing material of quartz, pyrite, and enargite. When this is the case, chalcocite as an intergrowth or later primary development, is universally present. The common form of appearance of bornite is a result of deposition from primary solutions, while these mineralizers are altering the previously formed vein minerals and forming new ones. Under these conditions bornite occupies an unstable position, readily yielding to copper enrichment or impoverishment of iron contents, to form the final mineral of the reactions, chalcocite. In contradistinction to this process of enrichment, bornite frequently forms in the impoverishment of rich ores, notably

covellite and chalcocite, as an intermediate product before the end reaction to chalcopyrite.

Chalcocite.—With the exception of some later chalcopyrite, chalcocite has been the last copper mineral to form in the Leonard vein, from ascending primary waters. Large and small stringers are seen running through solid, sericitized quartz-monzonite, as a direct replacement of the rock-forming minerals. Shattering of the country rock and vein minerals afforded channels for the entrance of mineralizers. Shots of the cuprous sulphide occur peppered through some areas of country rock. It is cer-

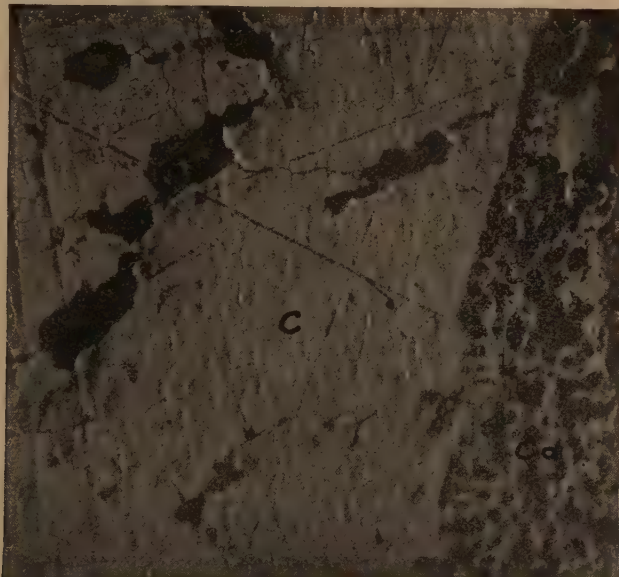


FIG. 12.—PRIMARY ETCHING STRUCTURE OF LEONARD CHALCOCITE REPLACING COVELLITE. $\times 80$.

C, chalcocite; Co, covellite.
1,600-ft. level, Leonard mine, Leonard vein.

tain that these agencies depositing glance were very strong in their effect on the quartz-monzonite minerals and those composing the veins.

Chalcocite is found traversing all of the earlier sulphides in veinlets and in replacement areas. Contacts of the earlier minerals and chalcocite are always sharp and distinct. With the exception of occasional structure derived from covellite, the chalcocite on etching produces a fine example of primary cleavage. The replacement of enargite by solutions depositing chalcocite has been described under enargite. This action has been universal and is far advanced in places. Other veins of enargite are untouched by chalcocite. Local conditions in the mineralizers, due to concentration of solutions, chemical composition, temperature, pressure,

and permeability of veins and wall rock, account for great developments of glance at some levels and minor amounts of it at no great distance in others. Bornite, covellite, enargite, and pyrite, represent the order of ease in which chalcocite replaces the vein minerals. Whole areas of bornite have been completely replaced. Advanced stages in the replacement of covellite prove that a complete transition took place from covellite to chalcocite. Similarly, enargite has been corroded until, as in the case of pyrite, rounded remnants alone attest the former crystals.

Fracturing after the deposition of covellite allowed access of solutions

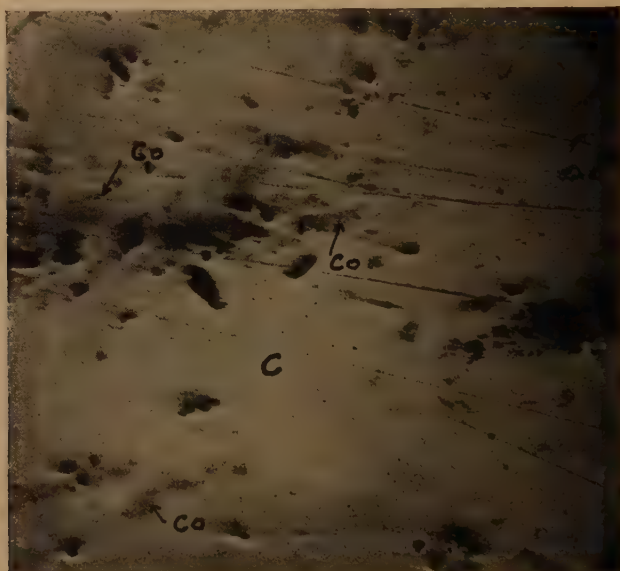


FIG. 13.—ALMOST COMPLETE REPLACEMENT OF COVELLITE BY CHALCOCITE. $\times 80$.
C, chalcocite; Co, covellite.
1,600-ft. level, Leonard mine, Leonard vein.

bearing chalcocite to the veins. (See Fig. 7.) Few lenses of covellite are found where glance has not to some degree been precipitated. When fine veins of chalcocite run through the covellite, they are found to develop prominent primary chalcocite structures, beautifully brought out by etching. When chalcocite grains are etched near covellite grains, they sometimes are found to contain shredded structures similar to that of covellite. Farther from the cupric sulphide the normal primary structure of glance is developed. (See Fig. 12.) This pseudomorphic etching of chalcocite is due to the minute shreds of covellite remaining wholly unchanged in the mass of cuprous sulphide. Prevalent mineral specks of bornite further complicate the field. Complete change to chalcocite usually, but not always, results in the assumption of its own crystal form.

Many areas of chalcocite that appear homogeneous before etching show large amounts of bornite and remnants of covellite after etching.

The primary character of this chalcocite is attested by crystalline intergrowths with bornite, primary intergrowths with the same and other minerals and characteristic primary structure developed by etching. Chalcocite from the deep levels of many other mines in Butte, gives the etch figures characteristic of the primary mineral and identical with those of the Leonard vein.

As already observed, the deposition of chalcocite occurred during one period of mineralization. The primary solutions carrying the cuprous



FIG. 14.—CHALCOPYRITE REPLACING COVELLITE AND CHALCOCITE. $\times 80$.

C, chalcocite; Ca, chalcopyrite; Co, covellite.
Leonard mine, Leonard vein.

sulphide first deposited bornite, with iron as the determining factor, until, by the reduction of the bornite molecules, the chalcocite began to precipitate as the earlier minerals were replaced, forming crystallographic intergrowths and primary interlockings with bornite. Finally chalcocite itself directly replaced the previously formed vein minerals and filled cavities, bornite forming at the same time in small amounts. In most of the observations on the occurrence of chalcocite it has been found to be a replacement of quartz-monzonite and vein minerals. Instances of chalcocite and quartz filling of vugs have been observed in the Gagnon mine and chalcocite vugs are occasionally found.

In the replacement of covellite, glance and bornite have played a very

intimate part. Covellite has altered directly to both bornite and glance, bornite being the first mineral formed as a result of this reaction. Later solutions not only replaced covellite and deposited glance, but also attacked and replaced the previously formed bornite. It is believed also, that at the time mineralizers were replacing covellite and depositing chalcocite in its place, a universal but finely divided development of bornite occurred. Fig. 13 shows a typical case of almost complete replacement of former covellite masses by chalcocite. Remnants of covellite can still be seen.

Chalcopyrite.—Chalcopyrite is present in the covellite in minor amounts. In fact it is seldom seen, and then only in microscopic grains, near crystals

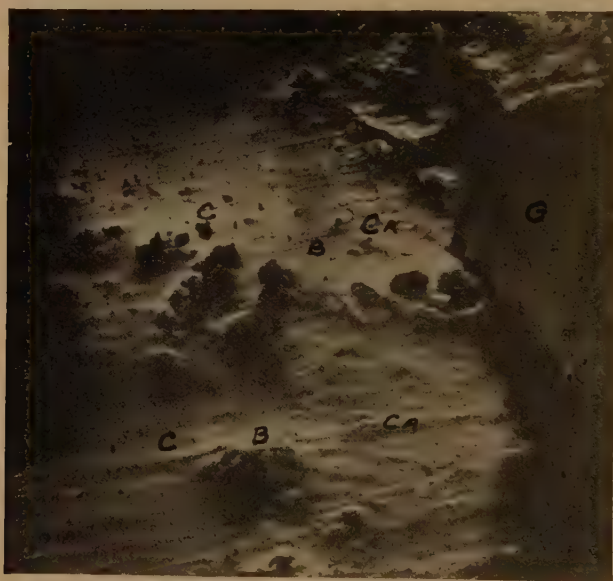


FIG. 15.—RIM OF BORNITE BETWEEN CHALCOPYRITE AND CHALCOCITE. $\times 80$.
B, bornite; C, chalcocite; Ca, chalcopyrite; G, gangue.
2,400-ft. level, Anaconda mine.

of pyrite or masses of bornite. It is a transitory, primary development from the action of solutions depositing bornite and chalcocite in the replacement of vein minerals. Slightly larger amounts of chalcopyrite are sometimes seen coating and replacing covellite crystals. In many cases covellite blades have been almost completely altered to chalcocite before the deposition of chalcopyrite. This chalcocite invariably gives a primary structure on etching and is considered to be a primary enrichment of the covellite. Chalcopyrite coatings on covellite crystals and more or less complete replacements of these crystals, present under the microscope somewhat conflicting evidence. It is believed that most, if not all, of this

chalcopyrite is primary, its reluctance to crystallize by itself complicating the structure. In Fig. 14 chalcopyrite has replaced covellite and chalcocite, but on account of the similar actinic properties of the minerals they can with difficulty be distinguished.

The action of pyrite and bornite on copper-bearing solutions is believed to bear a close relation to that of chalcopyrite. Most sections of ore minerals from the Leonard vein that contain chalcopyrite show almost invariably the chalcopyrite in close proximity to minerals containing iron such as pyrite and bornite. Chalcopyrite masses are, in such instances, believed to be the local result of the action of the later mineralizers on the iron sulphide. Chalcopyrite altering to chalcocite with an intermediate rim of bornite occurs on the 2,400-ft. level of the Anaconda mine. (See Fig. 15.) Linings of bornite surround some of the chalcopyrite of the Leonard vein. At other times the contacts of chalcopyrite veinlets are sharp and distinct.

3. *The Fault-Vein Covellite*

The Edith May vein in the Speculator mine, the Gray Rock vein in the East Gray Rock and Diamond mines, and the South Bell vein in the High Ore mine, have revealed covellite in scattered lenses from several inches to several feet in extent. All of these fault veins belong to the Blue vein system described by Sales.

The Speculator Ore.—Sections of the Edith May ore present a matrix of minerals very similar to those of the E-W. Leonard vein. Pyrite, enargite, tetrahedrite, covellite, bornite, and chalcocite, make up the ore.

Pyrite is present in crystals and in partly corroded grains. Being the first sulphide to form, it has undergone continued attack by later solutions. Enargite is found in irregular remnants, almost wholly replaced by covellite, tetrahedrite, and chalcocite. Practically all of the covellite is believed to be a replacement of enargite. Before the deposition of covellite, small but widespread quantities of tetrahedrite replaced enargite. Tetrahedrite is usually found in irregular small masses, not taking the form of veinlets. (See Fig. 16.) It is evidently a precipitation following enargite before the beginning of covellite deposition, although some areas have the appearance of intergrowths with covellite. Tetrahedrite strongly resists alteration.

Covellite contains many small particles of gangue. It is chiefly the result of the replacement of enargite, but also of crushed country rock and pyrite. The cupric sulphide is much mottled, partly by reason of a faint coloring of enargite that has almost completely altered to covellite. On etching with potassium cyanide, the enargite remnants are attacked more markedly than the covellite. In this manner some of the covellite is seen to retain the structure of the enargite it has replaced. A considera-

ble proportion of the mottled appearance is due to the alternation of covellite with a slightly lighter colored mineral, probably a lighter colored phase of the same mineral, as both varieties etch similarly.

Bornite occurs occasionally in veinlets through covellite and as a fringe along the contacts of chalcocite and covellite. It is an intermediate product of the alteration and is sparingly developed. Chalcopyrite, with the bornite, is the outcome of a local concentration of iron in the mineralizers, due sometimes to the presence of pyrite in the ore. Bornite is also found as shadowy specks in areas of covellite.



FIG. 16.—TETRAHEDRITE IN COVELLITE AND CHALCOCITE STRINGERS. $\times 80$.

C, chalcocite; Co, covellite; T, tetrahedrite.
800-ft. level, Speculator mine, Edith May vein.

In some sections the alteration of covellite has not begun. Others show that replacement by chalcocite has occurred to a slight extent. In all directions minute branching veinlets of glance, with sharply cut borders, are characteristic. Very little evidence remains of a shattering of the vein later than the deposition of covellite. In the replacement of the cupric sulphide by primary chalcocite, the sharp contacts of the veinlets, the crystalline etching structure of the chalcocite, and the absence of gradation of intermediate alteration products between the center of the veinlets and the covellite, signify its primary character. All of the minerals in this ore are undoubtedly primary. The development of tetrahedrite and a much smaller proportion of chalcocite, distinguish the ore from that of the E-W. Leonard vein.

The East Gray Rock Ore.—The specimens preserved of the ore minerals in the East Gray Rock vein show a uniform variety of constituents. The ore is a result of precipitation of minerals from solutions or igneous aqueous vapors, as they ascended through the country rock between strong clay walls of the vein and replaced the crushed material. As in the Leonard vein, quartz and pyrite constituted the first vein filling. Characteristic of this approach to the border zone between the copper and zinc veins as established by Sales, different minerals begin to appear. Sphalerite ranks next to pyrite in the paragenesis of the vein minerals, while covellite has universally replaced the gangue, pyrite, and zinc blende, at the same time precipitating directly from solution. Tetrahed-

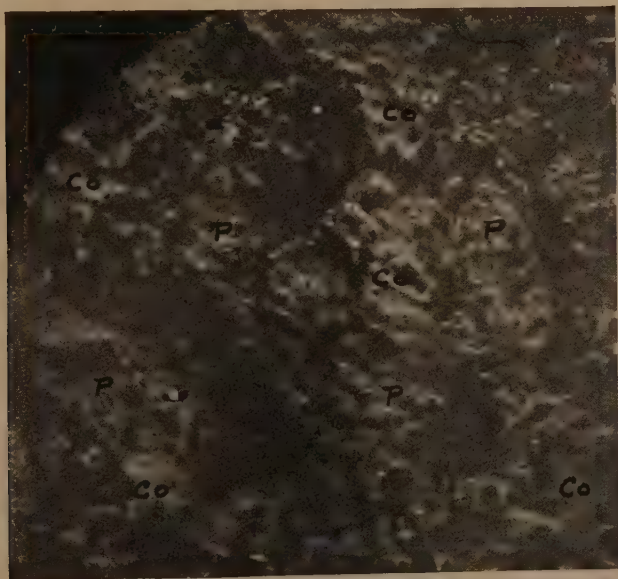


FIG. 17.—COVELLITE CEMENTING CRUSHED PYRITE GRAINS. $\times 80$.

Co, covellite; P, pyrite.
East Gray Rock mine.

rite is another mineral, uncommon in the Leonard ore, but abundant in the Gray Rock vein as an earlier mineral, an intergrowth with covellite and a later replacement of previously formed vein minerals. Minor amounts of chalcocite replace all other minerals, excepting tetrahedrite.

A feature prominent in the Gray Rock ore is the thorough shattering of pyrite and gangue before the deposition of covellite. Even minute pyrite grains show extensive shattering, the crystals being cemented and the cracks filled with covellite. In the course of enrichment, pyrite has been altered to several minerals, chief of which is covellite.

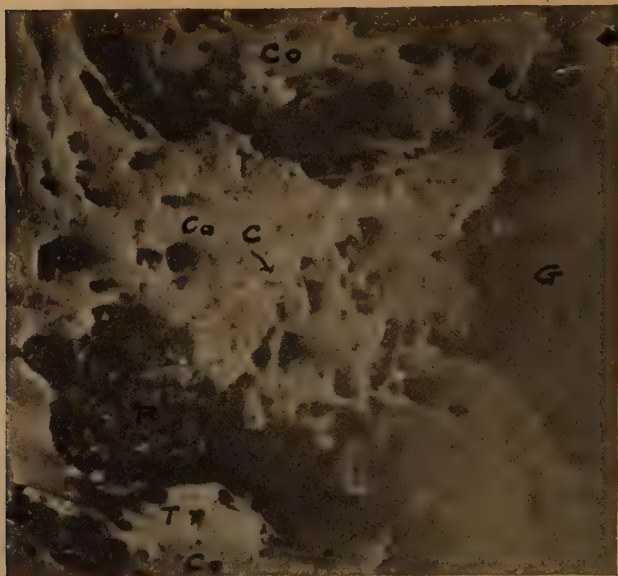


FIG. 18.—TETRAHEDRITE INTERGROWTH WITH CHALCOCITE AND REPLACING COVELLITE. $\times 80$.

C, chalcocite; Co, covellite; G, gangue; P, pyrite; T, tetrahedrite.
East Gray Rock mine.



FIG. 19.—CHALCOCITE REPLACING SHATTERED SPHALERITE. $\times 80$.

C, chalcocite; S, sphalerite.
East Gray Rock mine.

Many micro-crystalline specks of pyrite are peppered through the covellite, most of them remnants of former larger grains. (See Fig. 17.) An unusually well developed mottled structure of the cupric sulphide, caused by intergrowths of a light and a dark blue material, is found in this ore. The lighter blue portion of the mineral forms usually around inclusions of gangue. Etching develops characteristic covellite cleavage and a difference in the crystallographic orientation of blades of the mineral. Covellite has resulted from the replacement of pyrite, sphalerite, and gangue, and from a direct crystallization from solutions.

Features contrasting with the Leonard ore are: the absence of enargite; sprinklings of irregular small splotches of tetrahedrite; and little chalcocite. Tetrahedrite occurs as a deposition following sphalerite, as an intergrowth with covellite and, to a slight extent, as a replacement of it. It is generally intimately associated with gangue, as if clusters of quartz grains had favored its formation. (See Fig. 18.)

Chalcocite performs its usual function of replacing previously formed vein minerals. The apparent affinity of chalcocite for zinc blende has resulted in an advanced stage of replacement of the blende. (See Fig. 19.) Covellite also has totally replaced areas of sphalerite. The primary structure of the chalcocite, developed by etching, adds microscopic evidence to the structural proof of the primary nature of all of these sulphides.

In the rich ore occur exceedingly fine specks of galena, replacing the tetrahedrite.

4. Other Occurrences of Covellite

Rare instances of the formation of small bunches of covellite crystals have been met in numerous mines. The Mountain View and Tramway have been interesting in this respect. Pyrite crystals in Tramway specimens have been determined to be later than the covellite, and to be a filling between the blades. From the mines of the central copper area, the covellite, in general, is found to be altered in varying degree to chalcocite and later chalcopyrite.

Stopes in the High Ore mine, on the South Bell vein, a member of the Blue vein system, sometimes open up small clusters of covellite blades. These masses of the cupric sulphide are found to be replaced by chalcocite to an advanced degree, without the development of bornite or chalcopyrite. Remnants of pyrite crystals speck the copper minerals. Bunches of this material have recently been found, 2,400 ft. below the surface, in the South Bell vein. Included chalcocite exhibits the typical etching of primary chalcocite. The covellite of the fault veins may, in general, be said to contain more impurities than that of the E-W. veins, probably on account of the difference in solubility of the materials replaced in its formation. Insoluble impurities may be almost wholly referred to the greater

percentage of the rock-forming minerals only partly replaced in the formation of the fault-vein ore and the simultaneous deposition of silicates with the covellite.

Further from the central copper zone, even in the infrequent covellite specimens obtained, the presence of minerals typical of the intermediate copper zone is noted. Tetrahedrite and sphalerite are most frequently met. Covellite from the Bell mine, of the intermediate zone, contains small irregular masses of tetrahedrite, with branching veinlets, probably an intergrowth with the cupric sulphide and in part a later deposition. The mottling of covellite in this part of the district is pronounced. The replacement of a previously formed pyrite vein filling is universal.

III. CONCLUSIONS

1. *The Replacement of Minerals*

From microscopic evidence it is clear that the primary minerals of the veins have been subjected to the action of solutions penetrating them under conditions rendering the minerals unstable. It is also evident that the mineralizers entering the veins carried with them different elements, some of which, under the imposed conditions, were stable after reacting with the vein materials. Exchange of constituents of the mineralizers with those of the previously formed vein minerals resulted in the deposition of new minerals, stable under the prevailing conditions. Observed facts point to the interchange of chemical constituents of the minerals and later solutions and deposition of new minerals, occupying the same volume as those replaced. The process of replacement appears to have progressed under conditions of simultaneous dissolution and deposition, with the formation, in some cases, of minerals intermediate in composition.

Local conditions of replacement, that may be present only in the contact film between the solutions and the minerals, are responsible for varieties in mineralogical developments. Variable activity in circulating mineralizers has influenced chemical composition and volume changes in the process.

Evidence of replacement with no change in volume is found in patches of chalcocite in large areas of covellite. Here the solutions have penetrated covellite, adding copper or subtracting sulphur necessary to form cuprous sulphide. Covellite areas in pyrite grains, exactly filling the dissolved areas, chalcocite in pyrite grains and gangue, chalcocite in enargite crystals, all attest to the replacement by volume, the carrying away of mineral substance and addition of new material. Such replacement is believed to be due to slowly moving and arrested mineralizers.

Irregular veinlets of the later sulphides in the earlier vein minerals

chronicle replacement by volume. Further replacement has taken place around the edges of the earlier minerals, destroying surfaces and crystalline outline. No space of solution ordinarily can be observed, at the contacts of replacements.

Replacement of bornite by chalcocite presents sub-microscopic phenomena, which, on account of the intimate relations of the two minerals, vary from the common conditions of the process. Etching of primary chalcocite and bornite, in sections where covellite has been replaced by both, shows a good primary structure. The chalcocite etches with fairly uniform rectangular cleavage, the distinctive feature of its habit being the almost uniformly straight boundaries to the individual grains. Where bornite has been replaced the gradation of one mineral into another is sometimes very indistinct. The relations of these minerals are remarkable in this respect. Cleavage of chalcocite extends into grains of bornite, but is not so pronounced when pure bornite is encountered. A zone of incipient replacement develops good cleavage. It appears that the chemical interchange incident to the metasomatism of bornite, by solutions depositing chalcocite, bears a markedly close relation to the common molecular constituents of the two minerals and by the elimination of some molecules of Fe_2S_3 the percentage of chalcocite present is raised by successive gradation up to that of the solid chalcocite.

Such a type of structure is frequently difficult to interpret. Doubt may exist concerning the priority of minerals, but convincing proof is present in the fact that the metasomatism incident to the formation of bornite from covellite or enargite did not proceed to the replacement of the particles of gangue disseminated through the primary minerals. Thus the bornite contains remnants of gangue, while the action of the chalcocite mineralizers replacing bornite destroyed the included gangue. It is an observed fact, that in primary chalcocite a comparatively pure mineral may be expected in whatever quantity it may be present. Pyrite, the most common impurity in chalcocite, is more resistant to the action of the depositing solutions, but the presence of any quantity of gangue or ore minerals, excepting the intimately related bornite, is rare.

In replacement by primary ascending solutions different structures are involved from those of replacement by descending solutions. In the former case, the contacts of the minerals replaced and the minerals deposited are so sharp that microscopic examination reveals no solution line. If intermediate alteration products are formed, the same condition holds true. But in the case of alteration by descending waters, a hazy line of intermediate products with indefinite contacts, generally marks the zone of metasomatism. Such sharpness of outline in replacement by primary minerals is believed to be due to greater chemical activity of the solutions, a result of their chemical constitution, and the presence of high temperatures and pressures.

2. *The Character of Mineralizing Solutions*

Sales has outlined the probable condition of magmatic waters and igneo-aqueous vapors in their ascent through the fissures in the quartz-monzonite. He concludes that the earlier solutions were preponderantly alkaline, by reason, in some measure, of the subtraction of sodium, calcium, and manganese from the quartz-monzonite wall rock. Later solutions were less alkaline, since the altered wall rock and the deposition of quartz acted as an insulation for the solutions; and accordingly the percentage of hydrogen sulphide in the mineralizers increased. As a result of this change the final mineralizing solutions depositing chalcocite might be relatively poor in alkaline salts. Whatever was the constitution of these later solutions depositing chalcocite, it is certain that they were more active than the preceding mineralizers in replacing the most difficultly altered minerals. Final solutions rising in some parts of the veins were at times highly alkaline. The deposition of calcite resulted occasionally and that of barite commonly.

Paragenesis and Character of the Vein Minerals

The entrance of early minerals proceeded along previous fractures and altered the quartz-monzonite, forming solution channels and vugs. The greatest alteration took place along the main zone of fracturing.

Quartz and pyrite were the first minerals to be deposited, mainly as a replacement of quartz-monzonite. Enargite followed, and all three minerals continued to form. Covellite, chalcopyrite, bornite, and chalcocite followed in the order given.

Ferro-magnesian minerals of the quartz-monzonite are more readily attacked and replaced by solutions of the sulphides than the feldspars or quartz.

The same order of deposition is found in both E-W. vein ore and fault-vein ore, with the addition of tetrahedrite in the fault veins, of the same age and sometimes just previous, and continuing till slightly after the crystallization of covellite.

Quartz and pyrite depositing first were shattered and cemented by later minerals. Enargite shows little shattering, covellite generally a slight buckling of plates.

Covellite is a replacement mainly of enargite, other vein-forming and rock-forming minerals, and frequently crystallizes in the free space of vugs. In general, it strongly preserves its own crystal form, but sometimes shows internal structure after enargite.

Irregularity of the occurrences of covellite is regarded as due to local conditions in chemical constitution of the mineralizers, local physical conditions, chemical character of inclosing medium, arrested movement of solutions, and variation in temperature and pressure.

Slight shattering after deposition of covellite afforded channels for entrance of solutions depositing glance and bornite.

Mineralizers depositing chalcocite were very active in the replacement of previously formed vein minerals and country rock, more so than previous or later solutions. Chalcocite belongs to one continuous period of primary deposition and on etching is generally found to preserve its own internal structure. Smaller amounts of it occur in the covellite of the intermediate zone.

Bornite and chalcopyrite were developed with the entrance, through minute fracture planes and cleavages in earlier minerals, of solutions carrying the cuprous sulphide. Local concentration of iron in the mineralizers, sometimes due to vein pyrite, caused deposition of these two minerals.

The relation of bornite and chalcocite is very intimate. In the case of the Leonard ore, bornite formed before the chalcocite and continued to deposit simultaneously till enrichment ceased.

Bornite is very unstable and easily attacked, forming chalcocite and chalcopyrite. Bornite, covellite, sphalerite, enargite, pyrite, and quartz, represent the order of ease in which chalcocite replaces the vein minerals.

Tetrahedrite is present in the covellite of the intermediate copper zone but not in the central copper zone material, evidence formerly presented by Sales in describing change in character of mineralization outward from the central copper area.

Solutions of later sulphides have generally attacked and replaced earlier-formed vein minerals and quartz-monzonite.

In the alteration of pyrite to chalcocite an orderly succession of copper addition and iron subtraction occurs, to form chalcopyrite, bornite, covellite, and chalcocite.

The order of resistance to the alteration of primary sulphides by later mineralizers is: pyrite, tetrahedrite, enargite, covellite, chalcopyrite, bornite.

Differences in the distribution of minerals in the veins were probably due to highly local variations in the constitution of the mineralizers. The physical and chemical effects of the wall rocks and previously formed vein minerals were practically the same through large areas of orebodies where variations in mineralization occur.

Secondary bornite and secondary chalcopyrite are plentiful as associates of secondary chalcocite. They are frequently preserved as a halo around pyrite grains or residual as intermediate products in the descending secondary sulphide enrichment of pyrite.

The primary character of chalcocite is recognized under the microscope by eutectic patterns with recognized primary minerals, by primary intergrowths with the same minerals, by physical relations with primary minerals and by distinctive structure developed by etching.

The secondary nature of chalcocite is determined by decomposition cracks and characteristic etching structure.

The author is indebted to Reno H. Sales, Chief Geologist of the Anaconda Copper Co., for advice and criticism in the preparation of this paper and to Prof. D. C. Bard, of the Montana State School of Mines, for rare specimens and the loan of apparatus.

DISCUSSION

ARTHUR S. EAKLE, Berkeley, Cal.—At the old Morning Star mine in Alpine County, where large masses of enargite were found some years ago, I looked particularly for evidences of replacement of pyrite by the enargite. While I have not made a particular study of any of the specimens, I judge that the pyrite and enargite were precipitated together. They were both primarily in those rocks. Of course, by using Mr. Graton's method of determination we might possibly find a situation somewhat different, but I should say that the enargite is not replacing the pyrite, but was simply precipitated at the same time. The bodies occurred there within 50 ft. of the surface, so it is probably not a case of secondary enrichment.

H. W. TURNER, San Francisco, Cal.—In the case where secondary chalcocite shows a crystalline structure and cleavage lines, what is the evidence of the secondary nature of it?

C. F. TOLMAN, Jr., Stanford University, Cal.—You can find practically all the evidences we have discovered in the microscopic study of ores. You can find all stages in the process of replacement of earlier minerals, especially bornite, by chalcocite, from a stage, in which the process has but just started with only occasional fragments of chalcocite set in bornite, to a stage where remnants of bornite are set in chalcocite. The replacement may take place along the cleavage lines or along the crystallographic directions of the earlier mineral, or it may take place irregularly. In one type of replacement you may find in one spot a few cross-cutting veins, and in another spot the latter may be so enlarged that there remain only a few individual specks of the replaced mineral.

A. C. LAWSON, Berkeley, Cal.—I would like to ask whether this structure is really a pegmatite structure.

C. F. TOLMAN, Jr., Stanford University, Cal.—It is the so-called eutectic structure of Laney and others.

A. F. ROGERS, Stanford University, Cal.—In some cases the so-called graphic intergrowth of chalcocite and bornite is probably due to inherited structure from an earlier graphic intergrowth of bornite with

other minerals. In other cases it seems to be due to a partial breakdown of bornite into the chalcocite. I have not found any case where bornite and chalcocite were formed simultaneously. I may be mistaken but I do not believe that they were formed simultaneously from ascending solutions. The so-called intergrowth is a peculiar breakdown but it is rather difficult to know just what it means.

L. C. GRATON, Cambridge, Mass.—There was, as Professor Tolman has said (in his summary), an epoch-making paper on the application of the reflecting microscope to the study of sulphide ores, but that was the paper by Campbell. Similar methods had been employed many years earlier, but Campbell was the first to disclose the general applicability and importance of the idea and most of the progress that has been made in this direction is traceable directly or indirectly to the impetus he gave to this line of microscopic research. Murdoch and I were simply among the early followers of Campbell's methods, and if our paper contained any contribution it lay chiefly in our attempt at systematic attack on a single part of the entire field this new method opened up.

Apparently Mr. Thompson, and perhaps some others, have fallen into an error for which Murdoch and I may have been responsible, viz., the conclusion that a systematic pattern developed in chalcocite by etching with acid (cleavage structure, if you will), is a certain indication that the chalcocite was formed from ascending solutions, i.e., is "primary" in the generally accepted sense. Shortly after the appearance of our paper, Professor Tolman pointed out the danger in applying this conclusion too sweepingly, as we had likewise cautioned, but Mr. Thompson seems to have overlooked these warnings. There can be no doubt that under certain conditions a systematic structure revealed by etching is present in chalcocite that has been formed by descending solutions in the orthodox, secondary-enrichment fashion.

The question of the "graphic intergrowths" affords another topic for discussion. Laney was the first to note in the sulphides that peculiar pattern with which we are familiar in the pegmatites, and he concluded that it meant, in the one case, as had commonly been regarded in the other, simultaneous crystallization of the components. At the time our paper was published Murdoch and I had found no reason to dispute this interpretation, and I am by no means ready as yet to accept the extreme belief expressed by Professor Rogers that this structure never represents simultaneous intergrowth, though I am now satisfied that a very similar pattern may in certain cases result from partial alteration of one mineral to another.

Criticism of the terms "primary" and "secondary," particularly the latter, on the ground that they have more than one meaning, is to some extent justified. As Professor Tolman has indicated, the terms "hypo-

gene" and "supergene," proposed by Ransome, are valuable because of their precision as to direction from which the originating causes were derived. But the term "secondary enrichment" has been so extensively used and has come to mean so definite a thing, and to be understood so unusually well by all classes of people engaged in or connected with mining of the metals, that it would seem to be as futile as it would be unwise and confusing to endeavor to abandon it and substitute some other expression for it.

Equally to be deplored and avoided, it seems to me, is the application of the term "secondary enrichment" to results and processes that have little or nothing in common with those alterations by descending, meteoric waters that produce an increase in metal content of part of a deposit through the impoverishment of a nearby part. Coupling "upward" or "ascending" with "secondary enrichment" is in my opinion a mistake.

A. C. LAWSON.—One matter that appealed to me in connection with this discussion is the desirability of not disturbing the nomenclature after it has once been established in the mind of the layman or practical miner. If we change our nomenclature we will disturb people and confuse them greatly, and in fact we will undermine the confidence they have in our work. Whatever proposals are made for the change of the use of words such as "secondary" and "primary," we should always fully comprehend the meaning attached to those words by the miners who have come to use them. Those words to them have a certain definite meaning. I agree with Mr. Graton in the hope that the term "secondary" be retained, but that we should all have a definite understanding of the meaning of secondary enrichment. If the idea has prevailed that primary ores were all formed at one time I am very glad the trend of the discussion shows that that idea has been broken up.

J. C. RAY, Palo Alto, Cal.—I think I am probably more interested in Mr. Thompson's paper than any one else present, as I have written a paper on the same subject which appeared last year in *Economic Geology*. It naturally gives me no little satisfaction to find that Mr. Thompson departs rather cautiously from Sales' theory, and to note his practical verification of the principal points which I have already brought out; *i.e.*, that the Butte deposits were formed as the result of several distinct periods of mineralization whereby each set of sulphides belonging to one period was replaced by a later set of sulphides belonging to another period, and not, as thought by Sales, by a practically continuous circulation of a gradually changing solution which resulted in the simultaneous deposition of several sulphide ore minerals. I make this statement reservedly as I know that Sales wrote his paper before any detailed study of the Butte ores had been made with the use of the reflecting

microscope, and presume that he made statements then which he is willing to modify or change now.

The soundness of Mr. Thompson's criteria for distinguishing primary from secondary chalcocite by the etch structure has already been questioned. I also do not agree with Mr. Thompson in this point for I have seen much undoubted secondary chalcocite which exhibits the same regularity of etch structure which Mr. Thompson ascribes to primary chalcocite.

He also believes the transitional zones to be indicative of secondary or downward solutions. My own observations would lead me to place equal weight with this as evidence of minor side reactions which frequently accompany the replacement by ascending solutions. While undoubtedly taking place at times from the action of descending solutions I can not agree that it is a safe criterion for downward enrichment.

Mr. Thompson frequently refers to the replacement of quartz, pyrite and quartz-monzonite by primary chalcocite. I can not agree with this. I believe that any primary (deposited by ascending solutions) chalcocite is a replacement of earlier sulphides, enargite or sphalerite, usually through the intermediate stage of bornite. I believe that the replacement of pyrite by primary chalcocite is of little importance. The author mentions chalcocite occurring in veinlets up to 2 ft. in width which are direct replacements of quartz, pyrite, or monzonite. I have studied rich chalcocite ore from some of these little veins and find the chalcocite undoubtedly replaces earlier sulphides. One such, the south branch of the Minnie Healy, on the 1,600-ft. level of the Tramway mine, exhibits ore which is apparently pure chalcocite replacing pyrite and the monzonite wall rocks. Microscopic study shows this ore to be thickly dotted with residual fragments of enargite and bornite with occasional remnants of sphalerite. Some of this chalcocite is completely meshed with partially replaced enargite grains. At no place in Butte have I been able to find primary chalcocite which I could conscientiously ascribe to anything but a replacement of enargite, sphalerite, or covellite. The evidence is not always clear in one specimen, but by studying many specimens the solution of the replacement is sure to be found somewhere.

L. C. GRATON.—Do you exclude the bornite?

J. C. RAY.—By no means. In the area which is under discussion in Mr. Thompson's paper, chalcocite replaces enargite directly, but I am inclined to think that the greater part of it has gone through the bornite stage first.

H. W. TURNER.—I quite agree with what Professor Lawson has said in regard to retaining the terms "primary" and "secondary" in the original sense used by S. F. Emmons for the use of the mining fraternity. There is also a financial point in this. Investors in sulphide mines the world

over have come to know about primary and secondary ores, used in this simple sense; and they are interested because if a sulphide deposit has been leached above, it is nearly certain that some rich ore will be found below, unless the primary sulphides were merely barren pyrite. Even the intelligent miner now understands the significance of the two terms.

The distinction between the various zones in copper mines, in which the original minerals were chiefly pyrite and chalcopyrite, is very plain; but it is true that in complicated deposits, like those of Butte, the average miner, or even mining engineer, would not be able to draw altogether correct conclusions as to the limit of the zone of secondary enrichment and the primary zone below without the help of such investigations as those at Butte, by Emmons, Weed, Sales, Graton, Tolman, Rogers, Ray, Thompson, Simpson, and others. There was "upward sulphide enrichment" in the primary zone; and the original solutions may have reacted upon each other resulting in the replacement of one sulphide by another; yet all these reactions belong to the primary period, long antedating the later leaching and secondary sulphide enrichment period.

The scientific description and discussion of the reactions and replacements that took place during the primary period requires a special set of terms with definite meanings, but it is not necessary that the layman should be required to know them. This will be all the more true if investigators always add a chapter to their papers, in which their general conclusions are stated in plain language, and in which, in reference to ore deposits, the terms "primary" and "secondary" are used in their original sense. Moreover, the writer of a special paper owes it to the public, as well as to himself, to add such a special chapter, for, being more familiar than anyone else with the subject in hand, he can do this better.

Of paramount importance to the mining industry are the broad facts, in most sulphide deposits, of (1) a leached zone, (2) a zone of secondary enrichment, (3) a zone of primary ore, and the terms "primary" and "secondary" should be retained in this sense and so used not only in popular papers, but in scientific discussions in order to avoid confusion.

A. F. ROGERS.—We are accustomed to the use of the terms "primary" and "secondary" in a geological sense, but if the matter were looked into carefully, the word "secondary" would probably be found to have priority in a mineralogical sense. At any rate, the terms primary and secondary as used at present are ambiguous. Some of us use them in one sense and some in another. The difficulty is that sometimes in the same paper the author will use the terms in two different senses, so it seems to me that in a paper it is absolutely necessary to define the use of these terms. For my part, I should like to avoid the use of these terms altogether,

but that is a difficult thing to do. We could use such terms as "the first generation" and "the second generation." I favor the use of the terms "hypogene" and "supergene," as indicating the source of the solution.

The terms "primary" and "secondary" may then be dispensed with and in their place the words "hypogene" and "supergene" used. The point is that these terms indicate the source of the solutions, and that is the principal argument for their use.

C. F. TOLMAN, Jr.—There is a difficulty in regard to the present use of the terms "primary" and "secondary," which apparently has not been fully appreciated by the preceding speakers, but which is felt by those of us who are engaged in the microscopic study of ores. This difficulty is due to the fact that a genetic-geological meaning has been grafted on to the purely descriptive terms for those minerals formed at the expense of earlier sulphides. These terms must be divorced from any theory as to the origin of the ores or their structures; *e.g.*, I want to describe a certain ore, which is, say, a mass of bornite shot through with chalcocite stringers. I know that the bornite is being broken up and replaced by chalcocite, but I do not know anything as to where the solutions causing this change come from; whether they come up or down or sidewise or any other direction.

• Doctor Rogers suggests that we say chalcocite of the first and second generation. The difficulty here is that the first generation of some one mineral, say chalcocite, may be secondary in and derived from some earlier mineral, say bornite, and chalcocite of a second generation may be deposited in veinlets and therefore be primary in the mineralogical sense. In either case the chalcocite may be of the fourth, fifth or *n*th generation, if we broaden the term and consider each step or stage in the formation of minerals of the ore as a generation. Any of these different generations of minerals and stages of mineralization might be produced by ascending or descending solutions. In the microscopic study of an ore, we must have a set of names for those minerals that are formed at the expense, and by the destruction of (partial or complete) an earlier mineral or minerals. For such minerals geologists have used the term "secondary," long before the process of downward enrichment was discovered or the phrase "secondary sulphide enrichment" was invented. In the early usage, secondary carried no implication as to whether the solutions causing the alteration were hot or cold, ascending or descending. For example, lepidolite or muscovite develops along tin veins, as a secondary mineral. The solutions causing this alteration are probably in gaseous form and generally ascending. Two common secondary products in the alteration of feldspar are sericite and kaolin. The former is probably generally produced by ascending solutions and the latter by

descending solutions. Mr. Turner's suggestion, therefore, that we retain the terms "primary" and "secondary" in the *original* sense used by S. F. Emmons is somewhat confusing.

The speakers seem to be generally of the opinion that priority should not rule in this case; that secondary should carry with it the concept of oxidation producing a leached zone, of downward moving solutions, of wealth at or near the water level, of poverty with depth; to be, in short, a name for stock jobbers and engineers to conjure with, to entice the fearful or to frighten the sanguine. Since language is a living thing, ruled by force and necessity and not by logic and priority, these terms may finally acquire the burden of geological theories. If so, we must have a set of names for the microscopist. We must understand each other. Loose language produces loose thinking. Think of the harm to which the present confusion in nomenclature has given rise. Professor Krusch has said that he has never seen primary chalcocite. (He uses the term in its prior meaning.) His critics cited the "primary chalcocite" in Butte. This turns out to be secondary as Professor Krusch used the term, and only by a process of geological theorizing can it be called primary.

At the meeting of the Institute in February, 1914, there was a discussion¹ of the question "To What Extent is Chalcocite a Primary, and to What Extent a Secondary Mineral in Ore Deposits?" In this discussion J. D. Irving appears to use secondary in the geological sense. F. L. Ransome avoids the use of the terms and his remarks are not ambiguous. Waldemar Lindgren uses the term in the mineralogical sense speaking of primary chalcocite crystals, although taking part in a discussion in which most of the disputants are using the geological or a mixed meaning. L. C. Graton frankly assumes that if a mineral is secondary in a mineralogical sense, it therefore is formed by the processes of downward enrichment. It was this assumption that so greatly diminished the value of his epoch-making and otherwise valuable paper on *The Sulphide Ores of Copper*.²

It seems necessary, somehow, someway, to adopt a nomenclature accurately defined and understood by all. If, however, the struggle is to continue and a peace conference is not as yet in order, let each man state clearly the way in which he uses the terms "primary" and "secondary."

A. PERRY THOMPSON (communication to Secretary*).—Among the most interesting details of the study of opaque minerals in the sulphide ores are the criteria for distinguishing the primary from the secondary minerals and the relation of bornite to chalcocite. In the discussion,

¹*Trans.*, xlviii, 194 (1914).

²*Trans.*, xlv, 26 (1913).

* Received Oct. 18, 1915.

the statement was made that a regular cleavage or pattern developed in chalcocite by etching with acid is not an unquestionable indication that the chalcocite was formed by ascending solutions. In this I fully agree, as perfect cleavage structure developed by etching chalcocite deposited by descending waters has been observed in many Butte ores since the writing of my paper in March, 1914. In April, 1915, in a communication to the Editor, I attempted to bring the manuscript up to date and corrected the former conclusion that the regular cleavage lines developed by etching are a reliable criterion for the determination of primary chalcocite. Unfortunately, this correction arrived too late to be made in the paper.

The relation of bornite to chalcocite is difficult to understand at all times, particularly by means of the microscope alone. Most commonly the relation may be described, as Professor Rogers has said, as a peculiar breakdown of the bornite in the chalcocite. Tolman and Clark, combining an intimate knowledge of the ores through microscopic study with chemical investigation, have lately published an instructive paper on the formation of the sulphide ores, particularly those related closely to chalcocite. More recently in the *American Journal of Science*, June, 1914, E. H. Kraus published the results of chemical analyses of a number of minerals grading in chemical composition, in orderly fashion, from bornite to chalcocite. Many of the minerals described by Doctor Kraus are probably the opaque minerals observed under the microscope, having slightly different shades, but closely resembling bornite in all characteristics, that have puzzled the microscopist and have been referred to as products of the general breakdown of bornite.

Geology of the Burro Mountains Copper District, New Mexico

BY R. E. SOMERS, ITHACA, N. Y.

(San Francisco Meeting, September, 1915)

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I. INTRODUCTION

1. *Location, Topography, and Climate*

THE Burro Mountains are located in the southwestern part of New Mexico, in Grant County. The group is made up of two distinct mountain masses, known as the Little Burros and the Big Burros, the former lying about 6 miles southwest of Silver City, and the latter about 9 miles farther in the same direction.

They are separated by the valley of the Mangas River, a tributary of the Gila, and a stream only in name, except after a heavy down-pour of rain. The Little Burros attain an elevation of 6,625 ft. above sea level, and the Big Burros 8,054 ft., with the surrounding country averaging about 6,000 ft. To the south of these mountains stretches the desert plain, while to the north are the more frequent elevations of the mountainous country.



FIG. 1.—BIG BURRO MOUNTAINS FROM LEOPOLD, N. M. SHOWS A PORTION OF THE DISSECTED SLOPE WHICH EXTENDS FROM THE BIG BURROS TO THE MANGAS VALLEY.

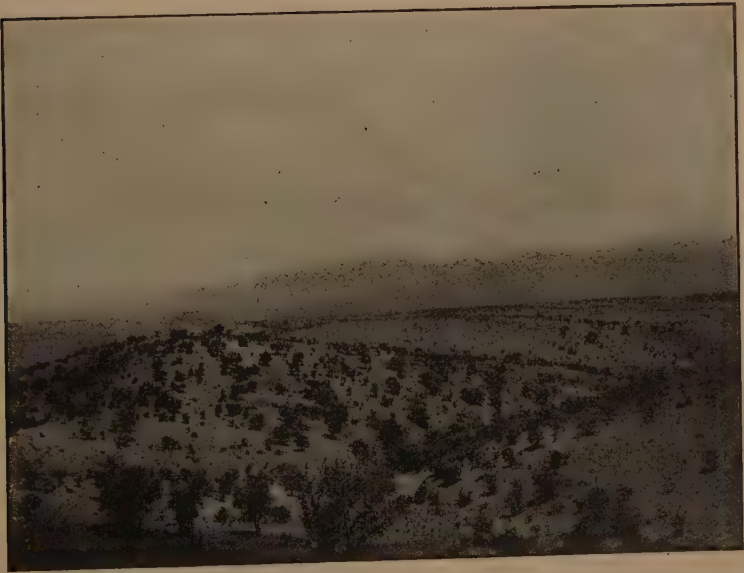


FIG. 2.—MANGAS VALLEY, SHOWING RIDGES OF PLEISTOCENE AND RECENT GRAVELS IN THE FOREGROUND.

The Little Burros are about 8 miles long, and $\frac{1}{2}$ to $1\frac{1}{2}$ miles wide, with a northwest-southeast trend. To the southwest they drop off rather abruptly into the Mangas Valley, but to the northeast their slope is gradual. The continental divide passes through the southeasterly end of this group, as well as through the summit of the Big Burros.

The Big Burros are more rounded in plan. They descend rather rapidly on the north and east from their maximum elevation of 8,054 ft. above sea level, down to about 7,000 ft. altitude, from which there is a rather gentle slope for about 5 miles until the surface drops off into the Mangas Valley on the northeast. This slope is made up of ridges separated by sharp gulches, or *arroyos* (Fig. 1), which are tributary to the Mangas. Near the foot of this slope is situated the camp of Tyrone, with Leopold about 2 miles farther up in a southwesterly direction.

The climate is arid, and hence the ridges are barren except for the sage brush, cactus, and mesquite, which detract little from the yellowish brown, or, in the mineralized section, rusty and jagged, appearance of the surface. In some of the gulches, however, a few evergreens and other trees may be found, especially about Tyrone, where in places the desert-like appearance is wholly lacking.

The accompanying map (Fig. 3) shows the relative location of the different points of interest in the district.

2. *Scope of Work and Acknowledgments*

The Burro Mountains, as a whole, contain two classes of ore deposits: (1) quartz veins with gold and silver; and (2) copper deposits in fracture zones with more or less disseminated replacement of the walls. The former have not been very extensively worked, but the latter have attracted considerable attention, and the richest of them, so far as shown by present development, lie in the vicinity of the two camps, Tyrone and Leopold. The object of this paper is to set forth the results of a rather detailed study upon this section of the Burro Mountains area.

The field work was done during the summer of 1913, and was made possible through the kindness of Dr. James Douglas and Walter Douglas of the Phelps-Dodge Co. To E. M. Sawyer, Superintendent of the Phelps-Dodge properties at the Burro Mountains, and to his staff, the writer wishes to extend his thanks for their hospitality and the facilities which they so willingly placed at his disposal. To Prof. Heinrich Ries the writer is particularly indebted for generous and thoughtful guidance and patient criticism throughout the work.

3. *History and Mining*

Copper was first discovered in 1871, and in the following years the district passed through various periods of activity and idleness



FIG. 3.—MAP OF THE BURRO MOUNTAINS REGION, NEW MEXICO.

until, in 1905, the Phelps-Dodge Co. secured the Burro Mountain Copper Co., which was organized in 1904 by the Leopold Brothers, of Chicago, to work a group of claims around the camp which is now called Leopold. In the vicinity of Tyrone there was another group of claims which was operated by the Chemung Copper Co., and in 1913 the Phelps-Dodge Co. purchased these also.¹

Since 1904, development in the Burro Mountains has been going on steadily and with excellent results. Tyrone is being made the



FIG. 4.—SAMPSON SHAFT AT LEOPOLD.

center of activity and it is reached by the recently completed Burro Mountains R. R., which runs from Whitewater on the Deming to Silver City spur of the Santa Fé.

At Leopold the principal shaft is the Sampson (Fig. 4), and through it a number of orebodies, such as the East and West Sampson, and the Protection, have been worked in the past. A short distance northeast of the Sampson shaft is a large deposit of developed ore, called the East orebody. There is also at Leopold the St. Louis shaft, an incline which is on the so-called "St. Louis slip," or fissure. South of the St. Louis is the McKinley shaft, which, however, has not been extensively worked. About $\frac{1}{2}$ mile northeast of Leopold are the Boston and Niagara shafts,

¹ For more details, see W. Rogers Wade: Burro Mountain Copper District, *Engineering and Mining Journal*, vol. xcvi, No. 7, pp. 287 to 289 (Aug. 15, 1914); also F. V. Bush: Burro Mountain Porphyry Copper Developments, *Mining Press*, vol. cx, No. 6, pp. 222 to 224 (Feb. 6, 1915).

neither of which has shown very much ore. The former connects with the Sampson workings. At Tyrone there are three shafts, called No. 1, No. 2, and No. 3. The first, which is separate from the other two, lies to the northwest and is not yet very important. Nos. 2 and 3 (Fig. 5) lie close together in the same gulch in which Tyrone is situated and open extensive deposits. Between the 300 and 400 ft. levels in No. 3 is a rich deposit in breccia, which is known as the "breccia orebody." Between No. 2 and No. 3 is the portal of the Niagara tunnel, a 7,000-ft. opening, running southwest, and extending to the East orebody at Leopold. It is connected by a crosscut with the Niagara shaft, and is



FIG. 5.—SHAFT NO. 2. ENTRANCE TO NIAGARA TUNNEL IS NEAR HOUSE IN LOWER RIGHT-HAND CORNER.

to be used to deliver the ore from the Leopold workings to the railroad, which is to enter it on grade. About 1,400 ft. from the portal, the Niagara tunnel has opened up a mass of ore called the "Bison orebody."

Besides the larger interests mentioned above, the Savannah Copper Co., Mangas Development Co., and others, are doing some work in the district.

II. GENERAL GEOLOGY

1. *Introduction*

The Big Burros, considered broadly, consist of a pre-Cambrian granite complex into which has been intruded a mass of quartz monzonite of post-Cretaceous age. The summit of the Big Burros is of

granite, and the flanks are either granite or monzonite down to the level where both are covered by the gravels of Pleistocene and Recent age. These gravels represent the débris worn from the earlier rocks and washed into the valleys or out on to the plain.²

The vicinity of Tyrone and Leopold includes a corner of this monzonite mass, and some of the granite into which it has been intruded. In detail, the rocks may be enumerated as follows: (1) pre-Cambrian granites; (2) quartz porphyry, of uncertain age; (3) post-Cretaceous quartz monzonites; (4) volcanic breccia of Tertiary age; (5) Pleistocene and Recent gravels.

2. Granite

Megascopic Characters.—The granites are medium- to coarse-grained rocks, containing rather large grains of feldspar when fresh, and abundant irregularly shaped grains of quartz. Biotite is occasionally seen but it is not common. The feldspars are of two kinds, a pink orthoclase, and a white plagioclase, and their colors may be easily distinguished in the hand specimen when the rock is not badly altered. Sometimes the feldspars are enough larger than the other minerals to give a slightly porphyritic texture.

The granite, in whatever condition it may be found, can be recognized megascopically by its abundant, irregularly shaped quartz grains, and by the lack of small feldspar phenocrysts. The first feature distinguishes it from the quartz porphyries, in that their quartz is in the form of rounded hexagonal phenocrysts and is not as abundant as in the granite; and also from the monzonite, which seldom contains quartz in large enough grains to be visible in the hand specimen. The granite is not sufficiently silicified to furnish any obstacle to this method of recognizing it. The lack of small feldspar phenocrysts in the granite distinguishes it from the porphyritic varieties of the monzonite as well as the quartz porphyry. The granite is never porphyritic with the exception of the very coarse phase already mentioned.

Microscopic Characters.—Under the microscope, the plagioclase proves to be an albite or oligoclase (Fig. 7). The orthoclase usually shows an exceedingly fine perthitic intergrowth with a plagioclase, which is too minute to be identified. In the granite as a whole, plagioclase is usually more abundant than orthoclase. Zircon, titanite, and apatite are present as accessories.

Structure, Distribution, and Age.—The granite is the original country rock of the district. It is found chiefly in the northern part of the area studied (see Fig. 6), where, however, it contains numerous and

² For a very good description of the Burro Mountains region, see Paige, Sidney: Ore Deposits near the Burro Mountains, New Mexico, *Bulletin* No. 470, U. S. Geological Survey, pp. 131 to 150 (1911).

irregular dikes of monzonite, which gradually diminish in size and number beyond the lateral limits of the map. These dikes are so irregular that they were not mapped in detail, but they are so numerous that this section of the area cannot be called granite without qualification.

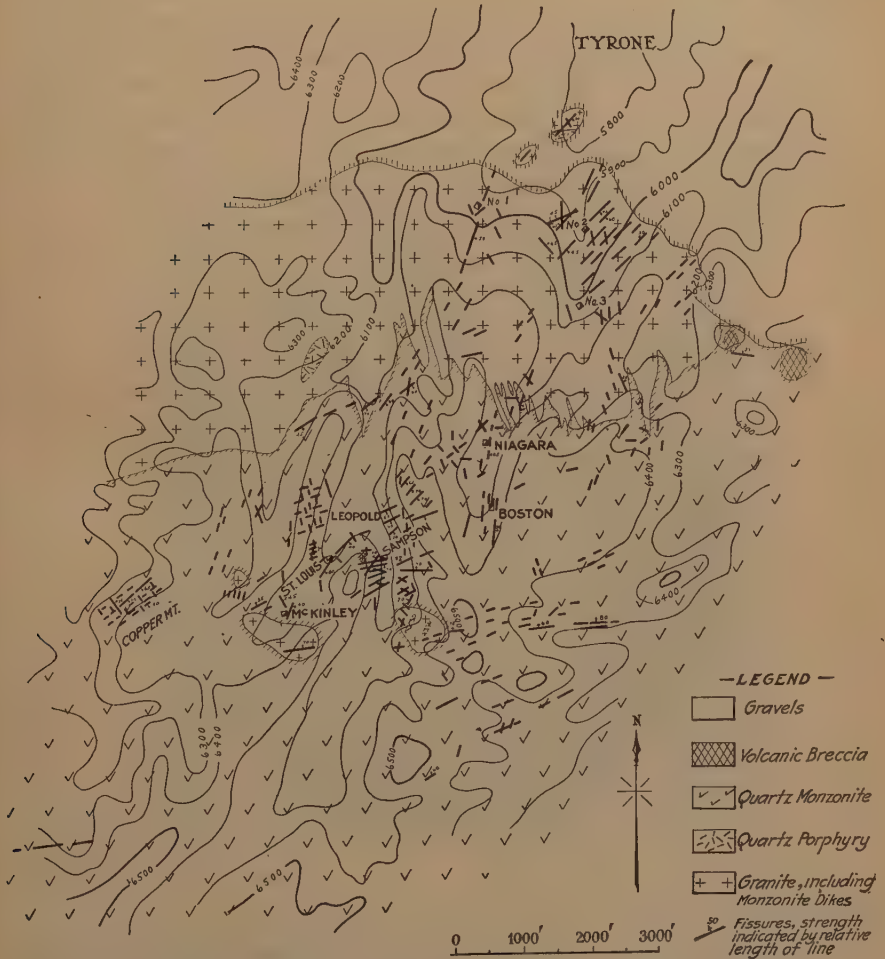


FIG. 6.—GEOLOGICAL MAP OF THE BURRO MOUNTAINS COPPER DISTRICT.

The granite included in this study of the Tyrone-Leopold district is but a very small part of the main mass. It is covered on the north by the Pleistocene and Recent gravels which fill the Mangas Valley, and does not appear directly opposite on the other side of the Mangas because of an intrusion of rhyolite porphyry which has cut through at that

point. It is found, however, about 4 miles north-northwest in the Little Burros.

In addition to this northerly extension of granite, there are several smaller areas in the central and southern parts of the district. About $\frac{1}{3}$ mile southwest of Leopold there are outcrops which extend 1,200 ft. in an east-west direction and 600 ft. north and south. Just south of Leopold is another area of about the same size. To the west of Leopold at a distance of a little less than $\frac{1}{2}$ mile is a third one of these "islands," but this occurrence is very much smaller than the first two mentioned, its greatest dimension being not more than 250 ft. Besides these, a single small outcrop appears 300 ft. northwest of the Sampson shaft, on the hill back of the old Burro Mountain Co. office.

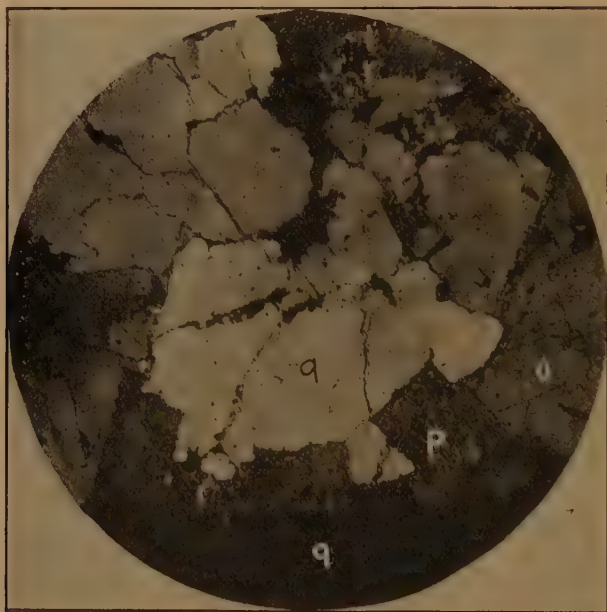


FIG. 7.—QUARTZ (q), ORTHOCLASE (o), AND PLAGIOCLASE (p) IN GRANITE.
CROSSED NICOLS. $\times 20$.

While slight textural and mineralogical variations are to be noted in the granite, there is no regularity in the distribution of the different phases. The age of the granite is pre-Cambrian, since it is found overlain by Cambrian sediments in some parts of the Silver City quadrangle.

3. Quartz Porphyry

Megascopic Characters.—The quartz porphyry is a light-colored porphyritic rock which always contains hexagonal phenocrysts of quartz,

varying in size up to 1 cm. in diameter of cross-section, and usually very striking phenocrysts of pink feldspar which are sometimes as much as 4 cm. long. The quartz phenocrysts show clearly their double-ended hexagonal pyramids, with edges somewhat rounded, and the pink feldspars have very well defined crystal outlines. In addition to these, smaller phenocrysts of a whitish feldspar are very common, and a few greenish grains of chloritized biotite were noted. In some places the white feldspars make up the background for the phenocrysts, but in others there is an aphanitic, blue-gray to white groundmass.

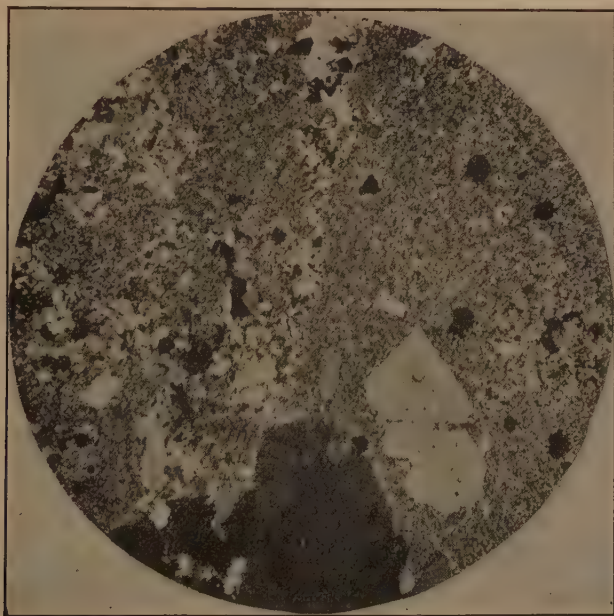


FIG. 8.—QUARTZ PHENOCRYSTS IN SILICIFIED QUARTZ PORPHYRY. TYPICAL UN-ALTERED SECTIONS SHOW MUCH MORE FELDSPAR. CROSSED NICOLS. $\times 20$.

Microscopic Characters.—Examined in thin section, the pink feldspar proves to be orthoclase, and the white, oligoclase. The notably rounded outlines of the quartz phenocrysts are clearly shown (Fig. 8). The groundmass is an aggregate of oligoclase, orthoclase, and an occasional grain of quartz. Apatite, zircon, and magnetite are the accessories present.

Structure, Distribution, and Age.—The quartz porphyry is intrusive into the granite. A large dike of it outcrops about 1 mile west of Leopold, just beyond Copper Mountain, and another one is found on the hill about $\frac{3}{4}$ mile northwest of Leopold, above the Leopold-Tyrone road. A few smaller dikes are found both underground and on the surface

in the vicinity of Tyrone, and in connection with the "islands" of granite to the south and east of Leopold.

The quartz porphyry is considered to be an earlier intrusion than the monzonite, since, while it is found in the granite, no occurrences have as yet been discovered where it cuts the monzonite. Furthermore, it is often mineralized. Its exact age is indeterminate, but the similarities in composition, structure, and texture between the quartz porphyry and the monzonite suggest the likelihood that the former did not precede the latter by a very long time, geologically, and that it was probably an earlier intrusion from the same magma.

4. Quartz Monzonite

Megascopic Characters.—The monzonites are extremely variable in texture. The most common phase about Tyrone and Leopold is a light-colored porphyritic rock of medium grain, containing abundant phenocrysts of white feldspar, and rarely a grain of quartz or biotite, in a dense whitish groundmass.

Some of the monzonite outcrops are of fine-grained porphyry rather than medium grained, and consist of a dense gray groundmass through which are sprinkled a few indistinct phenocrysts of feldspar.

Occasionally, however, there is found a phase which is almost granitic in texture. This contains an abundant whitish feldspar that shows striations and averages about 0.5 cm. in diameter, rather common grains of biotite which often show chloritization distinctly even in the hand specimen, some small grains of pinkish feldspar, and a little quartz which has the dull luster of an almost cryptocrystalline aggregate rather than the clear glassy luster of a single crystal. Whenever the texture of this rock tends toward porphyritic, the pink feldspar and the quartz make up the groundmass.

Different from any of these varieties is a rock which is found in narrow dikes in the granite in the northern part of the area. This rock is fine grained and dense, and wholly lacking in phenocrysts of any size. In the altered condition in which it is found, it is so quartzose and brittle in some occurrences that its resemblance to quartzite is marked. In other outcrops the quartzitic appearance may be less prominent, and at the opposite extreme are very dense phases which are so soft and smooth that they seem to be almost wholly lacking in quartz.

Microscopic Characters.—Under the microscope, the porphyritic varieties of the monzonite show large amounts of orthoclase and plagioclase, the latter, if anything, being slightly in excess. It does not predominate sufficiently, however, to change the identity of the rock. The plagioclase varies from oligoclase to andesine, the more basic phases being found to the south, away from the contact with the granite, which may

possibly be due to assimilation by the monzonite of blocks of the slightly more acid granite near the contact of the intrusive.

Quartz is present in practically all the monzonite, classifying it, therefore, as a quartz monzonite. It is very seldom present, however, in phenocrysts, but is rather to be found in small grains in the groundmass. Even when the texture of the monzonite is so coarse that it appears granitic in the hand specimen, the quartz is shown by the microscope to maintain its fine texture, filling in the interstices between the idiomorphic crystals of feldspar in the form of a fine aggregate, rather than in single grains that are nearly the same size as the feldspars. This feature is shown in Fig. 9 and is responsible for the rather dull luster of the

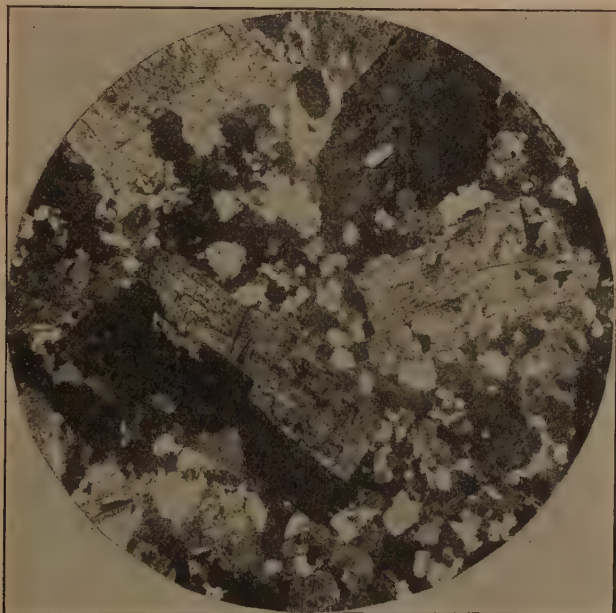


FIG. 9.—QUARTZ (SMALL GRAINS) AND FELDSPAR IN MONZONITE. CROSSED NICOLS. $\times 20$.

quartz as it shows in the hand specimen in these coarse varieties. It also furnishes a very good means of distinguishing the coarse monzonites from granite (Fig. 7).

The porphyritic monzonites contain apatite, zircon, and perhaps magnetite as common accessories. One section shows a small cluster of rutile needles.

The fine-grained dike rocks are found in thin section to be made up of quartz and secondary sericite (Fig. 10), no unaltered specimens having been obtainable. The phases which resemble quartzite in the hand specimen, show clearly in thin section an abundance of quartz in irregular

grains, together with many flakes of sericite. The quartz grains are intergrown with each other wherever they lie in contact. When they are not in contact, the spaces between them are filled with flakes of sericite, which form definite patches whenever the sericite is plentiful enough, and represent feldspars that have been altered. Quartz is relatively more abundant than the space occupied by the original feldspar, and even though some of it may be secondary, the rock as a whole was much more acid in character than the porphyritic monzonites.

The smoother phases are also made up of quartz and secondary sericite, but the patches of sericite are predominant and in extreme cases there is probably little or no primary quartz present. On the whole, their composition must have been more like that of the porphyritic monzonites.



FIG. 10.—FINE-GRAINED MONZONITE. GRAINS OF QUARTZ AND FLAKES OF SERICITE. CROSSED NICOLS. $\times 50$.

The texture of these dike rocks is exceedingly fine. In the quartzitic phases, about 0.13 mm. is the maximum diameter noted for a quartz grain, and the average piece of quartz is about 0.05 mm. in diameter. One section, with quartz very predominant, showed a flow structure, and an average size of grain of about 0.006 mm.

Structure, Distribution, and Age.—The quartz monzonite is the main intrusive in the granite. The exposure of it in the area mapped in Fig. 6 is, however, but a corner of the larger mass, which stretches away to the

southwest for about 6 miles, with an average width of about 3 miles. The intrusive nature of the monzonite mass is clearly shown by the dike-like character of some of its offshoots, the increasing fineness of texture in the monzonite as the contact is approached, and the slight baking which has been effected in the granite directly at the contact.



FIG. 11.—FINE-GRAINED MONZONITE DIKE (a) IN BADLY ALTERED QUARTZ PORPHYRY (b). NATURAL SIZE.

A part of the contact between the granite and this larger monzonite intrusion crosses the mineralized area (Fig. 6). It is characterized by tongues of the monzonite extending irregularly into the granite. Although the islands of granite about Leopold are mapped with rather

even contacts, it is probable that much more irregularity would be shown if the surface débris were sufficiently stripped to show bedrock. This is strongly suggested by such exposures as there are.

With the exception of the very fine-grained phase, the differently textured varieties of the monzonite show little regularity of distribution. The medium-grained porphyritic monzonite is the most abundant. A very coarse monzonite has been found in the Old Virginia shaft, about $\frac{1}{2}$ mile east of Leopold. At a similar distance south of Leopold the coarse unmineralized phase of the main mass first begins to appear. As any contact with the granite is approached, the finer variety of the porphyritic monzonite is usually found.

The dikes of aphanitic texture, which are found cutting the granite in the northern part of the area, require separate consideration. While these also intrude the quartz porphyry (Fig. 11), they seldom, if ever, cut the porphyritic or granitic monzonites. The writer believes that they represent that part of the monzonite magma which intruded the granite country rock in small dikes, either at the time of the main intrusion, or perhaps soon after, probably accompanied more or less by water vapor, and hence much after the fashion of the aplitic intrusions of a granite batholith.³ The name "aplite," however, is hardly applicable to all of these rocks. Many of them must, when fresh, have a composition very similar to that of the coarser monzonites, and they should therefore be called fine-grained monzonites. A few of the more quartzitic varieties may have the composition of true aplites, and they may be properly given the name aplite.

The age of the quartz monzonite is post-Cretaceous, as determined by Paige in his study of the Silver City quadrangle.⁴

5. Volcanic Breccia

Description.—This rock is a red porous clastic, consisting of fragments of varying sizes with 10 cm. as the maximum diameter. The fragments are quite angular and are composed mostly of red porous lava with occasional pieces of granite or monzonite. The breccia owes its red color to the fact that it is very thoroughly soaked with hematite and limonite which have been diffused by weathering of the lava fragments.

Structure, Distribution, and Age.—A few small outcrops of breccia are found about 1 mile south-southeast of Tyrone, near the road to the Savannah property. They are only a few feet thick and are the remnants of a formation which was once much more extensive than it is now. Other outcrops are to be found in the Little Burros. The outcrops observed to the south of the Mangas are stratified and have a northeast

³ Daly, R. A.: *Igneous Rocks and Their Origin*, p. 368 (1914).

⁴ *Bulletin No. 470, U. S. Geological Survey*, p. 133 (1911).

strike with a dip of 15° to the northwest. The pronounced red color in the vicinity is due to the large amount of iron oxides which the breccia contains.

The age of the breccia is presumably Tertiary.

6. Gravels

Description.—The gravels are made up of varying-sized fragments of all the older rocks of the district. The grains and pebbles are slightly rounded.

Structure, Distribution, and Age.—The gravels represent the débris worn from the other rocks of the area in Pleistocene and Recent times and washed into the valleys. Their distribution and rather angular form indicate their transportation and deposition by the few, but torrential, rains of an arid region, and this process is still going on.

They are sufficiently definite in form and composition to be a geologic formation, and they occupy the lower levels in the region, chiefly the Mangas Valley. Their southerly boundary is quite distinct and is shown on the map.

III. ECONOMIC GEOLOGY

1. Introduction

The copper of the Burro Mountains occurs in the mineral chalcocite, which has been produced by downward secondary enrichment. Chalcocite replaces primary pyrite, for the most part, and hence it is found, like the latter, as vein material, along with quartz, and as disseminated replacements in those portions of the rock bordering the fissures. The fractures, both large and small, which these minerals have filled, occur in a well-defined zone. Vertically, the three zones, of oxidation, downward secondary enrichment, and primary sulphides, are well marked, but their thicknesses and depths below the surface are very irregular.

2. Mineralogy

The primary vein and replacement minerals of the district are quartz, pyrite, chalcopyrite, and sphalerite. The secondary minerals resulting from (a) the processes of superficial alteration and downward enrichment are chrysocolla, malachite, azurite, cuprite, native copper, limonite, hematite, and chalcocite; (b) from hydrothermal action upon the rocks, sericite, chlorite, and epidote; and (c) from the action of descending meteoric waters on the rocks, kaolinite.

Quartz.—This mineral is not only present in the district as an original constituent of the igneous rocks, but it also occurs as a secondary mineral in three different forms, viz.:

(1) As a gangue mineral of the ores, where it is found in crystalline grains of moderate or small size in fractures, or saturating the country rock (Figs. 23 and 24). In this form it is either glassy or milky white, is found throughout the area, and has been deposited from ascending solutions.

(2) As a precipitate, separated out during the sericitization of the feldspars. This variety is not very abundant and is much less distinct than gangue quartz, but may occasionally be noted in thin section.

(3) As a weathering product, descending meteoric waters having reworked the earlier quartz more or less, and in some cases left it deposited in the upper portions of veins, as a crypto-crystalline aggregate, or chalcedony. This is often stained deep red with hematite, and in such condition may resemble jasper. It is found in the northern part of the district. The presence of hematite instead of limonite is interesting and corresponds with the observations of Emmons⁵ and others that hematite is apt to be the ultimate result of oxidation in place of a hydrous oxide, where the climate is arid.

Pyrite.—Pyrite is by far the most abundant primary sulphide of the district, and together with quartz makes up practically all of the original vein and replacement material. It is usually granular, and crystal faces or outlines are not very commonly seen. It is found throughout the mineralized area.

Chalcopyrite.—This mineral is distributed widely in the district but in very small amounts, and in such minute specks that it cannot, as a rule, be seen in the hand specimen, but its presence can be detected by microscopic examination of polished sections (Fig. 22). It may occur in pyrite or in quartz gangue, and is undoubtedly an original mineral. It appears to be very unevenly distributed.

Sphalerite.—Sphalerite was found in very few of the many specimens examined, and seems to be extremely rare, but, like chalcopyrite, it is present in such minute grains that its existence can seldom be noted except with the microscope (Fig. 17). There seems to be no doubt that it is of primary character, occurring with the pyrite and chalcopyrite.

Chrysocolla.—This hydrous copper silicate is the most abundant oxidized copper mineral found in the camp, and is chiefly responsible for the blue color of the outcrops at Copper Mountain, as well as at the Boston, No. 1, St. Louis, and McKinley shafts, etc. It not only occurs scattered irregularly through the surface portions of the rocks, but also may be found in small, clean-cut veins that are made up wholly of this mineral (Fig. 13). These are too free from quartz to represent former veins that have been changed to chrysocolla, but are probably later fractures in which solutions have deposited the hydrous copper silicate.

⁵ Emmons, S. F.: Los Pilares Mine, Nacozari, Mexico, *Economic Geology*, vol. i, No. 7, p. 633 (July-Aug., 1906).

The chrysocolla is colored various shades of blue and bluish green when pure.

A black, pitchy material is occasionally associated with the blue-green varieties of chrysocolla, as, for example, in the large fissure about $\frac{1}{8}$ mile southwest of Leopold. This contains manganese, and is probably copper pitch, or the German *Kupferpecherz*, a variety of chrysocolla stained brown or black with manganese oxide, which is well known at Globe.⁶ It is deeper black in color than much of the material from Globe.

Malachite.—Malachite is occasionally found in the district, though not in any abundance. At Copper Mountain it occurs in veinlets and coatings more or less intermixed with chrysocolla. It is fibrous, but the fibers are seldom more than 2 mm. in length.

Azurite.—Deep blue azurite occurs in very small quantities at Copper Mountain, but is rarely seen in any other part of the district. On the whole it is much more uncommon even than malachite. It is in small, irregular vein-like masses, and usually shows a thin streak of malachite running through the center of it.

Cuprite.—This mineral has been reported from the district,⁷ but did not come under the observation of the present writer.

Native Copper.—A few very small flakes of native copper have been found in monzonite in the Niagara tunnel about 3,000 ft. from the portal. They occur in very small and irregular fractures, and are usually accompanied by a red powdery hematite.

Limonite.—Limonite is very abundant as a surface stain and a filling of fracture spaces in the zone of oxidation. It is found throughout the mineralized area with the exception of a few localities where the oxidized form of the iron is hematite, and its quantity varies directly with the intensity of the primary pyritic mineralization. It shows a pronounced tendency to leach out on to the surface of the outcrops and talus fragments, thereby covering the surface with a thin brown film and giving the district its consistently rusty appearance.

Hematite.—In the northern part of the area, hematite seems to take the place of limonite to some extent in surface outcrops. It is red and finely divided and occurs either as a powdery material, or disseminated as a coloring matter in quartz. It is also found in considerable quantity in the volcanic breccia in the northeastern part of the district.

Chalcocite.—As at Butte, both massive and sooty varieties have been found in the Burro Mountains.⁸ No crystals, however, have been

⁶ Ransome, F. L.: *Geology of the Globe Copper District, Ariz., Professional Paper No. 12, U. S. Geological Survey*, p. 123 (1903).

⁷ Lindgren, Graton, and Gordon: *Ore Deposits of New Mexico, Professional Paper No. 68, U. S. Geological Survey*, p. 322 (1910).

⁸ Sales, Reno H.: *Ore Deposits at Butte, Mont., Trans.*, xlv, 49 (1913).

noted. The massive chalcocite is dark steel gray in color, and very homogeneous. It is by far the more common. The porous, sooty variety is occasionally seen where the massive chalcocite has been affected by meteoric waters in the upper parts of the chalcocite zone. Chalcocite shows well in polished sections (see Figs. 17 to 19). Its distribution is one of the main considerations of the district and will therefore be taken up under a later heading.

Sericite, Chlorite, and Epidote.—As will be discussed below, sericite, chlorite, and epidote have been developed by hydrothermal action upon the feldspars and ferro-magnesian minerals in the country rocks. The first is very abundant, the second rather uncommon, and the third rare.

Kaolinite.—Kaolinite has been formed from the non-metallic minerals of the rocks by meteoric waters, as will be noted in a later portion of this paper.

3. Occurrence of the Ores

The ores occur in two forms: (1) as the filling of open spaces, such as fissures of varying size and the interspaces of breccia; and (2) as disseminated grains in the walls adjacent to these open spaces.

(1) *Filling of Open Spaces.*—The district is characterized by a rather well-marked zone of fracturing, the surface outcrop of which is roughly triangular in form, with its apex toward the southwest, its sides on the west and southeast (see map, Fig. 6). Within this triangular area there are fractures of all sizes, from fissures that are prominent both on the surface and underground, to minute breaks that are visible only under the microscope. The fractured zone lies partly in granite and partly in monzonite, and shows some rather definite characteristics.

Larger Fractures.—The larger fractures, or “slips,” as they are called in the Burro Mountains, which vary from a fraction of an inch to a foot or more in width, are usually rather straight, and approximately parallel to each other over small areas. Their spacing varies from several inches to a few feet, and the dips are mostly steep.

The larger slips are often quite continuous and sometimes a single fissure may be followed for a considerable distance either on the surface or in the underground workings. More often, however, there appears to be a zone made up of two or more of these strong fractures accompanied by the usual large number of smaller breaks, and this zone may persist for 1 or 2 miles.

The slips at shaft No. 1 are good examples of large persistent fractures and all along the western side of the fractured area from this shaft to Copper Mountain the fissures form a well-marked, though somewhat irregular, zone. About $\frac{1}{2}$ mile southwest of Leopold there is a strong slip which is part of a zone that can be traced for nearly a mile across the country in a direction about 10° north of east. Likewise the southern-

most fissure plotted on the map is a part of a distinct zone extending to the northeast, and readily traceable on the surface from one end to the other.

It is an important fact, however, that only along the sides of the area do we find fissures or fracture zones of one general strike, with comparatively few and unimportant cross fractures. In the center of the area, as, for example, around Leopold and Tyrone, there are several fracture zones, which cross each other at various angles. These fissures are fully as strong as those along the boundaries, but not quite as persistent. Furthermore, some of them are not as straight, such as the so-called "St. Louis slip," for instance, which is shown at the St. Louis shaft. Southward from the shaft it has a general northeasterly strike, but on the north it curves until it has an easterly strike, which it holds until it has crossed the gulch in which Leopold is situated, when it turns back to a northeasterly trend. The most important direction of cross fracturing is roughly north and south, as for example in the immediate vicinity of Leopold; and again, near the Niagara and Boston shafts. The broad result of the fracturing, then, seems to have been a generally broken condition along the center of the area grading into a few consistent fissures along the sides. The latter do not cease abruptly as the distance from the center increases, but gradually diminish in size and number until solid rock is reached.

Two important structural characteristics of the fractured zone are defined by the larger fractures. In the first place, there is an apparent convergence of many of the slips toward an imaginary point about 2 miles southwest of Leopold. This may be noted on the map and is most pronounced toward the lateral boundaries of the area, where cross fracturing is uncommon. It is less marked in the center, where fissures of several systems are present, but even there it appears to be a strong tendency. On the north and northeast the gravels cover the continuation of the fissures, and thereby conceal their structure in that direction. At the eastern corner of the fractured zone, however, the fractures seem to be branching and fraying out, and it is quite likely that this structure is continued to the north.

Another noteworthy tendency is in the directions of dip taken by many of the slips. If the angle which the fractures make by their convergence be bisected, most of the fissures in the western half that are not vertical dip toward the east and southeast, while those in the southeast half show a tendency to dip to the northwest. This, of course, refers to the majority, but nevertheless the exceptions are comparatively few.

It is seldom possible to say whether or not there has been movement along the fractures. Since the rocks are massive, the displacement of strata cannot be used as a criterion, and recourse must be had to the less often observed evidences of displacement of dikes and cross fractures,

presence of gouge, and slickensides. Most of the fractures show none of these, and there is no indication of a big fault such as might cut off an entire rock formation, or leave a marked escarpment on the surface. On the other hand, some faulting is undoubtedly present. Slickensides are occasionally seen, a good example being in an outcrop on the road just south of Tyrone. Gouge may be found in a few slips, the St. Louis for instance. Small displacements of fissures may be observed, as in one of the Leopold orebodies, where such movements were noted as a 1-in.



FIG. 12.—LIMONITE-FILLED VEINLET IN FINE-GRAINED MONZONITE, SHOWING SMALL DISPLACEMENT. NATURAL SIZE.

fissure displaced 6 in. Very small displacements are shown in hand specimens from shaft No. 1 (Fig. 12). The evidence, therefore, points to the existence of small and irregular faults, perhaps in abundance, with big movements very doubtful and as yet unproved in any given instance.

Another question is in regard to the relative ages of fissures of different strikes. That some of the fractures are younger than others is clear in many instances. The small displacements noted above are cases in point. In the writer's opinion, however, generalizations regarding

the different directional systems of fractures are impossible. No one system is distinctly older or younger than another.



FIG. 13.—VEIN OF PURE CHRYSOCOLLA IN MONZONITE. NATURAL SIZE.

Smaller Fractures and Breccia.—Besides these larger breaks with their more or less consistent strikes and dips, there are very many small fractures, never more than a few inches in length, and often the merest frac-

tion of an inch in width. They have no regularity in direction or length. Their abundance depends on the brittleness of the rocks, and while in most of the monzonites they appear simply as frequent hair-like cracks, in the brittle granite they are much more abundant, and sometimes have developed to such an extent that breccias are the result. The breccia fragments vary in size up to about 1 ft. for the largest dimension, and may be found in small masses or in large bodies of lens-like form several hundred feet in length. That the brecciation is due to the brittleness of the granite is well shown by the fact that dikes of monzonite passing through a mass of breccia, though affected by the same stresses, are not fractured to the extent of brecciation.

Brecciation has caused at least one very important orebody in the district, which has been opened up between the 300 and 400 ft. levels in the shaft No. 3 workings. In addition to this occurrence breccia is found at one or two other points underground, and also at several places on the surface.

Origin of the Fractures.—While at best only a speculation, nevertheless it may be worth while to formulate a theory for the origin of these fractures.

They could not have been caused by contraction of the monzonite on cooling, since they are found only along a small portion of the monzonite mass.

They may possibly belong to a shear zone, but this seems rather improbable, for two reasons. In the first place, a shear zone of such great width and length would be expected to show much greater evidences of displacement than are found in the Leopold-Tyrone area. In the second place, it is very difficult to imagine a closed shear zone in which the fractures in each half dip toward the center. A fairly consistent dip in one direction would be expected.

The writer believes that this fracture zone can be explained by assuming a caving of the surface due to the extrusion of molten material from immediately beneath. There has been igneous activity all about this district in fairly recent times, and especially later than the monzonite there has been the intrusion of stocks of rhyolite which are found directly across the Mangas Valley from the fractured area.⁹ The fracturing was caused by the movement of this rhyolite, and probably commenced some time before the rhyolite became solidified in its present position. It continued, however, after the solidification of the rhyolite, since a few fractures are found in the latter, and their northeast strikes indicate that they are continuations of the fractures to the south of the Mangas.

With this hypothesis as a basis, the characteristic features of the fractured area can be readily accounted for.

⁹ Paige, Sidney: *Bulletin No. 470, U. S. Geological Survey*, p. 133 and map (1911).

A diagrammatic cross-section of such an area is shown in Fig. 14. In the first place, the fractures along both sides of the area would not be vertical, but would dip toward each other, since caving involves an area of surface larger than that of the cavity into which the rocks have fallen. In mining operations it has been found that in hard rocks the outermost fractures dip at angles of from 60° to 75° , and these correspond with the angles of dip noted in the vicinity of Leopold and Tyrone. The central fractures would show more nearly vertical dips.

While the majority of the larger fissures might dip as described above, there would, nevertheless, be many cross fractures and fracture zones, which would break the rock into blocks, especially in the center of the caved area. Along the edges the tendency might be to develop fissures in a single direction, but toward the center a generally broken condition

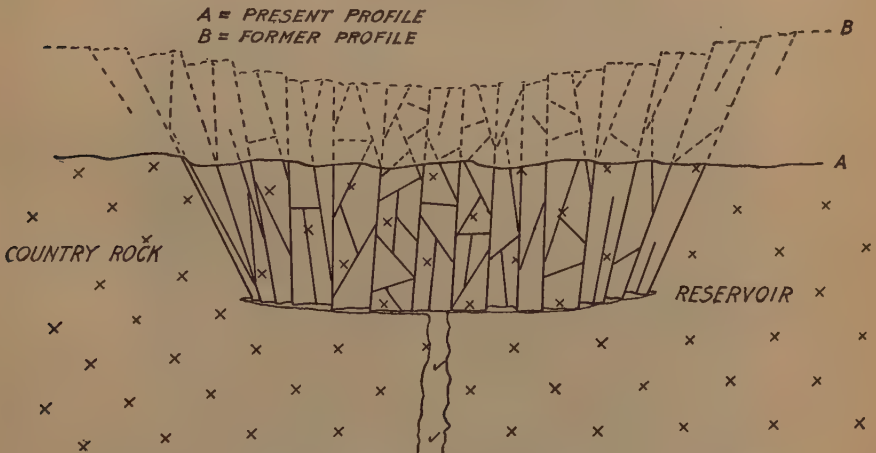


FIG. 14.—DIAGRAMMATIC CROSS-SECTION OF FRACTURED ZONE.

of the rocks, with more fractures, and more directions of fracture, would be produced.

Furthermore, innumerable smaller fractures would be developed, increasing in number with greater brittleness of the rock, until in extreme cases a breccia might result. This is the explanation of the breccia found at Tyrone, and the brittle rock is the granite, while in the tougher monzonite breccias are seldom, if ever, found.

Finally, the actual caving would consist of a settling of the blocks formed by these breaks and a readjustment to the new conditions, which would cause a small amount of displacement along nearly all the fissures, without relatively great movement in any case. And while all the fractures would not be of exactly the same age, on the other hand, no consistent differences in age between different systems of fractures would be expected.

Mineralization of the Fractures.—In these fractures, solutions have deposited quartz, pyrite, chalcopyrite, and sphalerite, and secondary processes have enriched portions of them to an extent sufficient to make their mining profitable. But in addition to the ore which is found in this multitude of fractures, there is an important amount found disseminated in their walls.

(2) *Disseminations*.—Primary pyrite has been deposited in the walls of the visible fractures in grains which range in size from exceedingly small particles up to those which are perhaps 5 mm. in diameter, with about 1 mm. as the average.

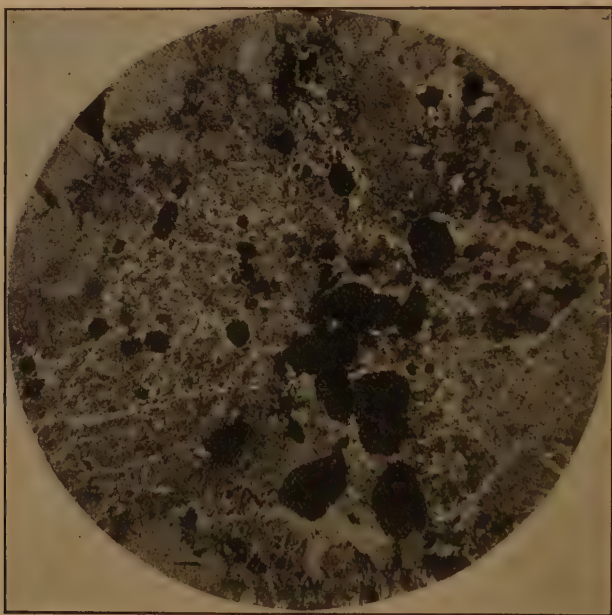


FIG. 15.—PYRITE (BLACK) REPLACING SERICITIZED FELDSPAR. LIGHT STREAKS ARE SERICITE. QUARTZ GRAINS AROUND THE EDGES. CROSSED NICOLS. $\times 50$.

Upon examination of thin sections of the ores, it is found that these grains were deposited in two ways: (1) as filling of minute open spaces; and (2) as metasomatic replacements.

The former occupy the irregular fracture spaces of microscopic size that exist in the rocks, and are of course similar in mode of origin to the more evident open-space fillings, the distinction between them being simply an arbitrary difference in size.

These micro-fractures develop commonly between the mineral grains, and also throughout the feldspars, as in the case of the granites, but to a much less degree in the quartz. The same is true in the mon-

zonites, but with the quartz more subordinate the fractures are more evenly distributed through the rock.

Secondary quartz is often present with the pyrite, and thin sections show all stages of gradation, from the smallest fractures with only one or two grains in them to the more distinct veinlets, where the filling is a granular aggregate of quartz and pyrite.

The second class of disseminations, however, have been formed strictly by metasomatic substitution, and they appear under the microscope to be for the most part replacements of the feldspars (Fig. 15). Clearly



FIG. 16.—PYRITE (BLACK) REPLACING QUARTZ OF THE GRANITE. SHOWS FRACTURES THAT SERVED AS CHANNELS OF ACCESS. CROSSED NICOLS. $\times 50$.

refined cubes of pyrite are often developed, and some grains show a connection with small fractures which have served as channels of access.

Many grains show the characteristics of both types, in that they are fracture fillings extended by replacement. Each of these occasionally appears to replace the quartz as well as the feldspars (Fig. 16).

The disseminations are scattered through the walls of the fissures for varying distances, depending upon the openness of the fracturing and therefore the freedom with which circulating waters could penetrate the rock. The walls are usually more or less impregnated from one slip to the next, and therefore the actual distance can seldom be measured.

4. *Oxidation and Downward Enrichment*

The value of these deposits as a source of copper is due to the presence of a definite, though irregular, zone of secondary chalcocite of sufficient extent and richness to work.

Oxidized Zone.—Between the secondary chalcocite and the surface there is a zone of oxidation which is irregular in depth, and from which the copper has been very thoroughly leached over most of the area. The pyrite which formerly existed in the fractures and disseminations has been changed to limonite, and the rock itself has a bleached appearance.

The depth to which leaching has extended is extremely variable. The fissures are leached much deeper than the less pervious rock between them, indicating that the freedom with which surface waters may reach considerable depths is a big factor in determining the vertical distribution of the different zones. It is not at all uncommon to find the slips thoroughly leached while the disseminations in their walls are enriched.

Furthermore, steep fissures are apt to be leached deeper than the flat ones. It is occasionally found that where a flat slip crosses a steep one, the latter is leached, whereas the former is enriched, which the writer would attribute, not to any difference in age between the two, but rather to the fact that a steep fissure would allow freer and deeper circulation of surface waters than would a flat one, with consequent deeper leaching.

In a few places some copper remains in the oxidized zone. It is in the form of chrysocolla and malachite, with some azurite and rare occurrences of cuprite and native copper in very small amounts. Such is the case at shaft No. 1, at the Boston, St. Louis, and McKinley shafts, and at Copper Mountain, as well as in a few other scattered areas. With the possible exception of Copper Mountain, there is hardly enough oxidized copper at any place to be workable.

The occasional presence of copper minerals in the oxidized zone is probably due to the impermeability of the rocks in these localities, which has hindered a free downward circulation of meteoric waters, and thus forced the precipitation of copper within the zone of oxidation, chrysocolla having been the result to a much greater extent than malachite or azurite because of the scarcity of carbonic acid, and the presence of silica under conditions suited to the formation of the silicate.¹⁰ The cause of this impermeability is not always clear. It may be due to the tightness of the original fracturing, or to an unusual amount of silicification. In a few cases it would appear that dikes of the more impermeable quartz porphyry have been responsible.

¹⁰ Kemp, J. F.: Secondary Enrichment in Ore-Deposits of Copper, *Economic Geology*, vol. i, No. 1, pp. 24 to 25 (Oct.-Nov., 1905).

Enriched Zone.—Below the oxidized zone is a zone of secondary enrichment in which the mineral developed is chalcocite.

The chalcocite is metallic or dull in luster in most cases, though occasionally sooty where recent changes in water level have allowed oxidizing waters to come in contact with it and by partial solution convert the massive chalcocite into the porous, sooty form.¹¹ Although the ores have been thoroughly examined under the microscope by means of reflected light, no secondary copper sulphides, other than chalcocite, have been disclosed.

The chalcocite has been developed by replacement of primary pyrite, chalcopyrite, and sphalerite. Sphalerite is rare, but nevertheless, as shown in Fig. 17, it is easily replaced by chalcocite. Although this has not been as generally observed in the studies of other ore deposits as has the replacement of pyrite and chalcopyrite, it is not at all unknown, and is clearly shown in the Burro Mountain specimens.

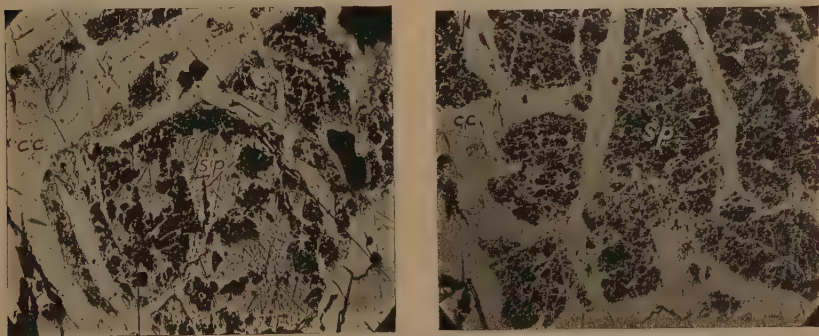


FIG. 17.—SPHALERITE (sp) BEING REPLACED BY CHALCOCITE (cc). BLACK SPOTS ARE PITS IN THE SURFACE. REFLECTED LIGHT. $\times 40$.

A complete gradation may be observed, from the first formation of a film about the original sulphide grain, with a thin veinlet or two running through it, to the point where there is nothing left but an occasional small residual core in a matrix of chalcocite, which indicates with shadowy outlines the size and shape of the replaced pyrite grains (see Figs. 18 and 19). No sulphides intermediate between pyrite and chalcocite have been noted.¹²

Since it therefore occupies the position of the primary sulphides, the chalcocite is found in fissures and fractures intergrown with quartz, and in disseminated grains in the adjacent walls (see Fig. 20). The ore as it is blocked out will average about 2 per cent. at Leopold and 3 per cent..

¹¹ Sales, Reno H.: *Trans.*, xlv, 49 (1913).

¹² Graton, L. C., and Murdoch, Joseph: *The Sulphide Ores of Copper*, *Trans.*, xlv, 40 (1913).

at Tyrone. The slips are much richer than this, but the poorer rock between brings down the value of the portion to be mined.

The chalcocite zone is irregular in thickness, with a maximum of 200 to 300 ft. It is found throughout the horizontal range of the mineralized area with the exception of one or two places where imperviousness of the rocks has caused a diffusion of the copper-bearing waters and consequent lack of a secondary zone.¹³ About 175 ft. southeast of shaft No. 2 is an instance.

The zone of enrichment does not show a close relation to the present water table. At the time of the writer's visit, this level was at a depth of from 275 to 400 ft., with the Leopold orebodies to a considerable extent above the ground-water level; while at Tyrone large masses of chalcocite are both above and below it.

Taken as a whole, however, the secondary zone shows a very regular

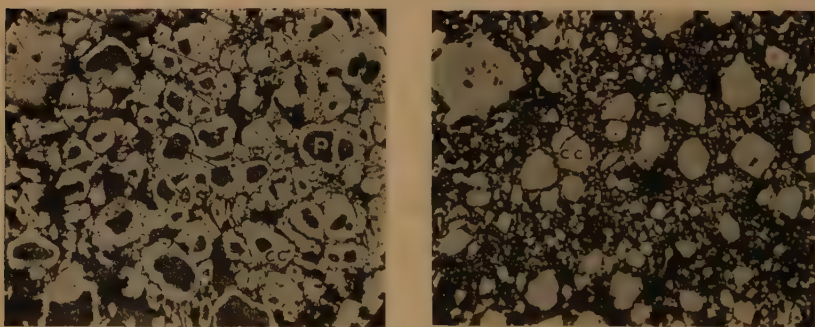


FIG. 18.—REPLACEMENT OF PYRITE (p) BY CHALCOCITE (cc). OUTLINES OF ORIGINAL PYRITE GRAINS INDICATED BY HOLLOWES AND GANGUE MINERALS, WHICH APPEAR BLACK. REFLECTED LIGHT. $\times 40$.

arrangement in the matter of depth from the present surface. It is quite near to the surface in the southern part of the area, as, for example, at Leopold, but becomes gradually deeper toward the northeast. When the Sampson shaft was sunk, chalcocite was found within a very few feet of the surface, and even along the larger fissures oxidation has not extended very deep. At Tyrone, however, the secondary zone is found at a considerable depth, and leaching has gone as deep as 700 ft. in places. Observations at intermediate points prove a gradually increasing depth from Leopold to Tyrone.

This tendency of the secondary zone, and its lack of correspondence with the present water table are due to the changes which have taken place in the relief, and consequently the water level, of the district.

¹³ Penrose, R. A. F.: *Economic Geology*, vol. ix, No. 1, p. 20 (Jan., 1914); also Emmons, W. H.: *Bulletin No. 529, U. S. Geological Survey*, p. 20 (1913).

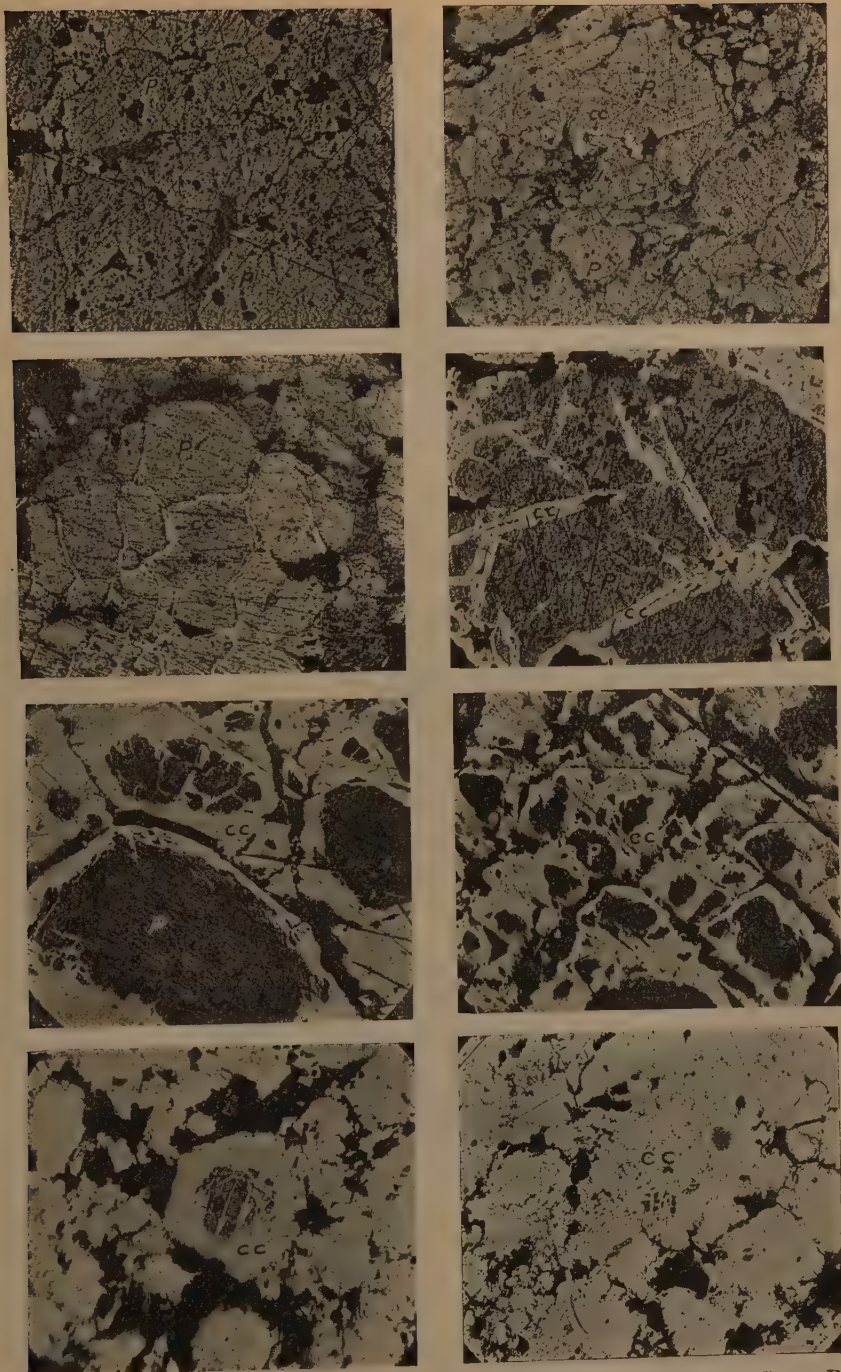


FIG. 19.—PHOTOMICROGRAPHS OF POLISHED SPECIMENS, ARRANGED TO SHOW PROGRESSIVE STAGES IN THE REPLACEMENT OF PYRITE (p) BY CHALCOCITE (cc). REFLECTED LIGHT. $\times 40$.

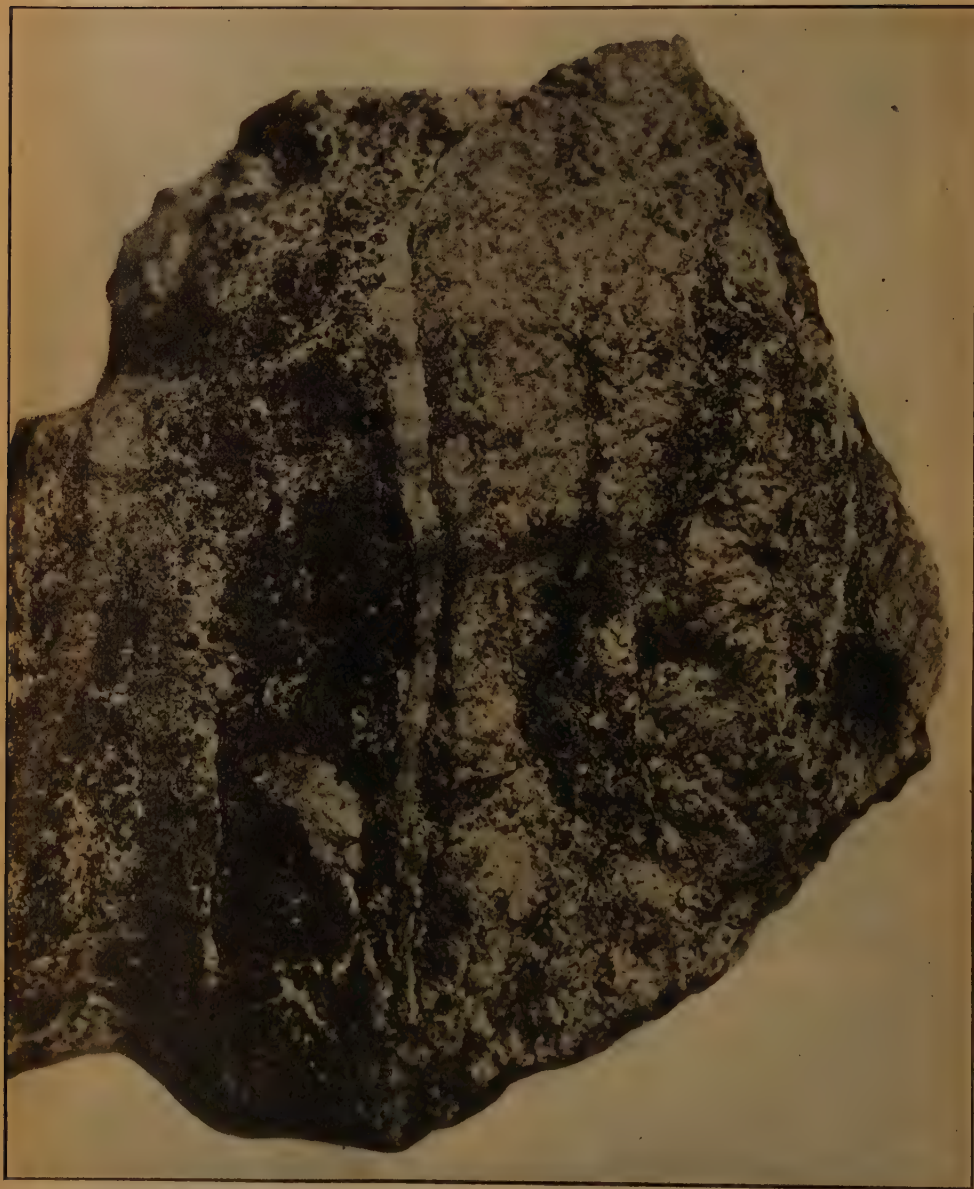


FIG. 20.—SPECIMEN OF ORE FROM SAMPSON SHAFT. SHOWS “SLIP” WITH LATER VEINLET OF QUARTZ. BLACK IS CHALCOCITE, WHITE IS QUARTZ. NATURAL SIZE.

In Fig. 21 the heavy line represents a profile from the Big Burros to the Mangas Valley, and on it are indicated the locations of Leopold and Tyrone. The broken line, *C*, represents the general profile of some former surface before erosion had worn the land down to its present level.

The location of this second profile is based on the tendency of erosion to wear down hills or mountains faster than valleys, especially in an arid region where so much of the work of erosion is done by infrequent but torrential rains that wash débris from the heights down into the larger valleys, but do not maintain constant flowing streams to erode these valleys very rapidly.

The Mangas Valley itself was probably initiated during a former period of erosion when the climate was less arid than it is now, or perhaps in the early part of the present period, under somewhat different conditions than those which prevail to-day. But for a large part

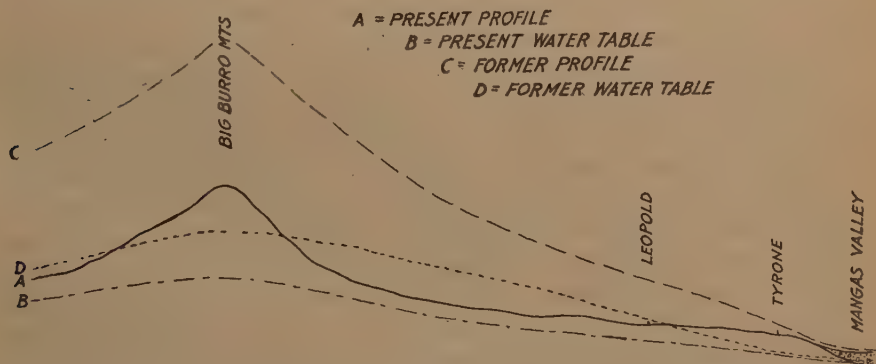


FIG. 21.—PROFILE OF BIG BURRO MOUNTAINS COPPER DISTRICT.

of the present stage of erosion the climate has been arid, as is proved by the angular form and undecomposed condition of the gravels, and the Big Burros have been worn down relatively fast, while the Mangas Valley has remained about where it was.

In this connection it is notable that the outcrops of many fissures about Tyrone contain, in place of the limonite and granular quartz which are found in the southern part of the area, a chalcedonic or crypto-crystalline variety of silica stained deep red with hematite. Chalcedony¹⁴ is a mineral that, under these conditions, would form very slowly, at or near the surface, and its presence in the more northerly veins indicates that their outcrops have been nearly stationary for a long time, whereas its

¹⁴ *Bulletin No. 529, U. S. Geological Survey, pp. 156 to 157 (1913).*

absence to the southward indicates that in this section the veins have been wearing away constantly without any part of them remaining long enough at the surface for chalcedony to develop.

Likewise, hematite stains this crypto-crystalline silica, but limonite is the surface form of the iron to the southward, which indicates that the northern outcrops have been longer exposed to the influence of the dry, hot climate than the more southern ones. These facts serve as additional evidence in favor of the second profile of Fig. 21.

Finally, the dotted line, *D*, represents the water table of such a former surface as that shown by the line *C*.

The important feature of the diagram is that the former water table crosses the present surface profile about at Leopold, but gradually diverges from it toward Tyrone. In a similar manner, any other former water level would pass comparatively near to the surface under Leopold, but at a considerably greater depth under Tyrone, and only recently has the water table been depressed below Leopold, while it has been as far below Tyrone as it is at present, during this whole stage of erosion. Hence, if the secondary zone is formed at or near the water level, it should be found at a depth of several hundred feet below Tyrone, because the water table has been at that depth for a long time. But at Leopold conditions have been different. Although the secondary zone tended to form at the water level, the latter has been sinking so fast that the former has been unable to keep up with it. This is because, in an arid climate, run off, with its accompanying erosion, is large, but absorption, and consequently alteration and enrichment, are small, so that erosion and the lowering of the water table go ahead of the downward migration of the secondary zone. In this way the chalcocite zone at Leopold has been left marooned above the water table and very near to the present surface. Going from Leopold to Tyrone, the water table has been considerably below the surface for an increasingly longer time, so that a better opportunity for leaching has been given and the depth of the chalcocite zone below the present surface gradually increases, while the amount of downward migration of this zone becomes less.

In the above discussion it is considered that the secondary sulphides could not have been precipitated consistently above the water level, because, on the whole, the fracturing of the Burro Mountains district is open and abundant, and there must have been a free enough access of oxygen and air to prevent any such precipitation.

Primary Zone.—The secondary sulphide at depth gives way to unaltered primary minerals. The line of separation between the two is neither regular nor sharp, and the depth at which it is reached depends on the permeability of the rocks. Locally, primary sulphides are found within a few feet of the surface, and unaltered cores are quite common throughout the deposit. On the other hand, the deepest workings, which

reach about 300 ft. at Leopold and 720 ft. at Tyrone, probably represent the lowest limit of any possible enriched zone.

Source of Copper.—The primary sulphides contain copper, although in very small quantities, and from the standpoint of a few years ago it would have been sufficient to say that the original pyrite was “cupriferous,” without further qualification.

More recently, however, the work of Simpson at Butte¹⁵ and the broader studies of Graton and Murdoch¹⁶ indicate that, chemically, cupriferous pyrite does not exist, but that the copper in lean pyrite masses is present as a copper mineral scattered in very small amounts through the other sulphides.

The primary copper of the Burro Mountains furnishes an example of this. Chalcopyrite is found in polished sections of primary sulphides taken from the Bison orebody and from the workings of the Sampson, No. 1, No. 2, and No. 3 shafts. It is generally very scarce, is usually too small to detect even with a hand lens, and undoubtedly represents the source of the copper.

It occurs in small irregular grains, in veinlets, and in more or less rounded patches (Fig. 22). It is believed to be approximately contemporaneous with pyrite. When in grains, chalcopyrite and pyrite may occur as neighbors, with interlocking contacts, and neither mineral impressing its form upon the other. When in veinlets they may occur together in a similar manner, or they may be found in different parts of what appears to be one fracture. While the patches of chalcopyrite in pyrite at first seem to be quite rounded and drop-like, upon further examination they show irregular outlines that indicate contemporaneous formation. Very similar patches are found in the quartz.

Specimens from some orebodies will show much more chalcopyrite in the primary sulphides than others, although there seems to be no regularity to this variation. It is safe to conclude, however, that an extensive examination of primary sulphide specimens would disclose regional differences in the primary copper content, which in some cases might well account even for the presence or absence of chalcocite.

Surface Ore-Indicators.—The surface of the mineralized area, as a whole, differs from that of the rest of the Burro Mountains region in color and form of outcrops. The limonite and hematite, which have leached out on to the surfaces of the outcrops and rock débris, stain them red-brown, and thereby give the area a generally rusty appearance. Furthermore, the outcrops are more rough and jagged than in the unaffected parts of the district.

Considered in more detail, there are three fairly reliable surface

¹⁵ Simpson, J. F.: Relation of Copper to Pyrite in the Lean Copper Ores of Butte, Mont., *Economic Geology*, vol. iii, No. 7, p. 628 (Oct.-Nov., 1908).

¹⁶ Graton, L. C., and Murdoch, Joseph: *Trans.*, xlv, 42 (1913).

indicators of ore, viz.: (1) the presence of fissures; (2) exceptional rust, either in the outcrops or in the rock débris; and (3) pyrite replacement cavities.

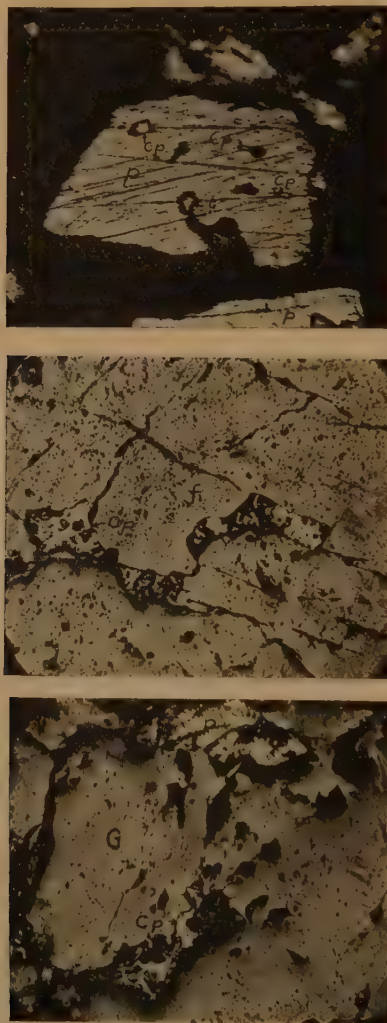


FIG. 22.—CHALCOPYRITE (cp) IN PYRITE (p), FELDSPAR (f), AND QUARTZ. GANGUE (G). REFLECTED LIGHT. $\times 40$.

(1) Fissures.—The ore-bearing fissures outcrop occasionally at the surface, although silicification has been hardly strong enough to enable the majority of them to withstand the weathering agents. When, however, fissures do appear at the surface, they may indicate ore if they are persistent, of sufficient width, and of more than one direction of strike. Without persistence, or a fair width or openness, an occasional

outcropping fissure would hardly indicate a zone of fissuring, and it is the latter in which orebodies are found in the Burro Mountains. The requirement that cross fissures should be present is perhaps open to question, but nevertheless it is doubtful if the solutions which deposited the primary minerals, and the meteoric waters which converted them into ores, were able to work effectively except in zones where a free circulation was promoted by the general breaking up caused by fissures in more than one direction. As noted above, the good orebodies of the district have been found in places where cross fracturing is common. Some ore may be found where cross fracturing is absent, but it will probably not be very plentiful.

In some places, especially about Tyrone, the presence of large fissures is proved, not by the outcrops, but by fragments of vein filling, mixed in with other surface débris, in such locations that it could not have been transported very far.

(2) Exceptional Rust.—Especially large amounts of limonite or hematite at the surface show indirectly that mineralization has been more extensive than usual. This indicates ore, provided that chalcopyrite was a constituent of the primary minerals, and that there has been sufficient fracturing to allow surface waters to form an enriched zone. The former could not be proved from any surface evidence, and if chalcopyrite is unevenly distributed, as seems probable from a study of the district, herein lies a difficulty in locating ore from the surface showing. It is not an all-important difficulty, however, because the cases where chalcopyrite seems lacking are the exceptions rather than the rule.

Nearly all the orebodies of the district show exceptional rust in their outcrops. The new Bison orebody is well indicated by a rusted area which is very noticeable from any of the surrounding hills.

(3) Pyrite Replacement Cavities.—Cavities which, from their size and shape, must have formerly contained pyrite, are also valuable indicators of ore. They are subject to the same qualifications, however, as given above in reference to the presence of limonite and hematite.

They are to be found above most of the orebodies, and are of special importance over the shaft No. 3 workings, where they seem to be about the only signs that show the location of the ore.

5. Rock Alteration

Chloritization.—At a distance from the fractures, beyond the sphere of intense mineralizing action, and where pyrite is very seldom developed, the biotite of the rocks has been changed to chlorite, epidote, and magnetite. This is equally true of monzonite, granite, and quartz porphyry, but results in small quantities of these secondary minerals because of the relative scarcity of original biotite. Other ferro-magnesian minerals

share the same alteration, but they are present in such very small amounts that consideration of them is unnecessary.

As seen in the hand specimen, the biotite thus altered may show merely a greenish color and a lack of elasticity in its plates where the change has not proceeded far, but the final product is a greenish powder, without semblance to micaceous form.

Under the microscope all stages may be observed, from the earlier, where cores of fresh biotite are abundant, through those in which no biotite remains but the outline of the original grain is still clearly defined, to the stages where the chlorite seems to have diffused into neighboring cracks as well as occupying the volume of the primary mica. This last stage produces a very irregular mass of alteration products, which shows only an indistinct resemblance in form to the original biotite.

Rocks which are altered in this manner contain small amounts of sericite.

Sericitization.—In the fractured area, however, sericite is very abundant. It replaces the feldspars in very minute flakes and may possibly develop in quartz in extreme cases.

Plagioclase changes to sericite earlier than orthoclase. The alteration proceeds gradually from the first stage, where a few scattering flakes of sericite appear in the feldspar, to the final product, which is a densely packed aggregate of sericite flakes with occasional grains of the secondary quartz which results from this alteration.

As seen in the hand specimen, the effect is a whitening of the feldspars with the destruction of their luster and cleavage planes. The silky character of the sericite is frequently evident, but it is not general, due to the fineness of the sericite flakes.

Not only has sericite been formed at the expense of the feldspars, but also it has replaced the chlorite of the earlier stage of alteration. In both slides and hand specimens may be observed the large flakes of sericite which occupy the positions once held by the chlorite. The flakes of sericite thus developed are much larger than those formed in the feldspars.

Sericitization is most pronounced immediately adjoining the ore-bearing fractures, and gradually diminishes as the distance from these fissures increases. It is also found intermingled with the quartz-pyrite filling of the fissures, where it has been carried by solutions from the walls.

Sericitization affects all the intrusive rocks of the district. It is perhaps most strikingly developed in some of the fine-grained monzonites which occur as dikes in the northern part of the district. Those phases of this rock which contained very little primary quartz have been converted by this process, combined probably with some kaolinization, into clay-like masses which soften when wet and in many ways resemble

fault gouge. This clay-like material is probably made up of sericite, secondary quartz, and kaolinite, and the intrusive character of its parent rock is proved beyond a doubt by the fact that it often shoots off tongues or branches from the main stringers.

Silicification.—Silicification of the wall rocks has taken place in the Burro Mountains to varying degrees (Figs. 23 and 24). In many places the rock on each side of the fractures is saturated for short distances with secondary silica, which replaces the original minerals, and, at times, may completely obliterate its texture. In some places, however, there is practically no such saturation, the secondary quartz being confined

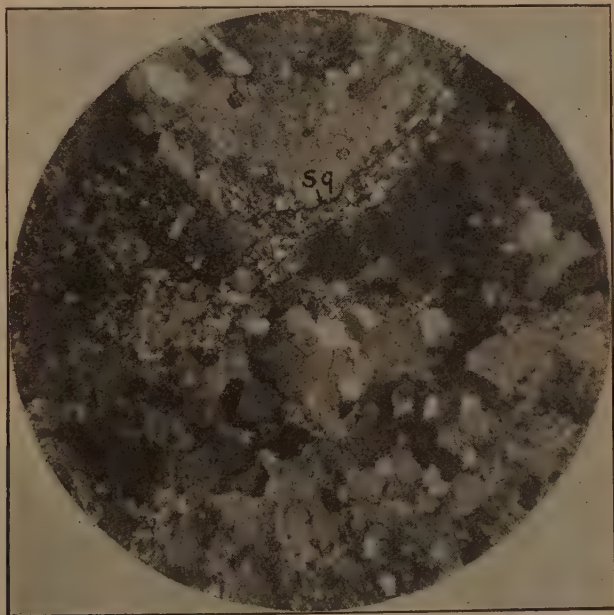


FIG. 23.—VERY FINE SECONDARY QUARTZ (sq) IN MONZONITE.
CROSSED NICOLS. $\times 50$.

to the small fractures, with perhaps an occasional replacement grain. On the whole, silicification has not been nearly as abundant as in many deposits of similar type.

When present in large amounts, secondary silica, with the accompanying iron oxide, gives a strong gossan-like character to the outcrops. These may be noted in any part of the area (Fig. 25). Although these seem rather common in a survey of the district, nevertheless they do not represent any very large proportion of the total number of fissures.

Study with the microscope suggests that there were at least two generations of secondary silica, one of which accompanied the primary sulphides, while the other followed at a later time and filled fractures

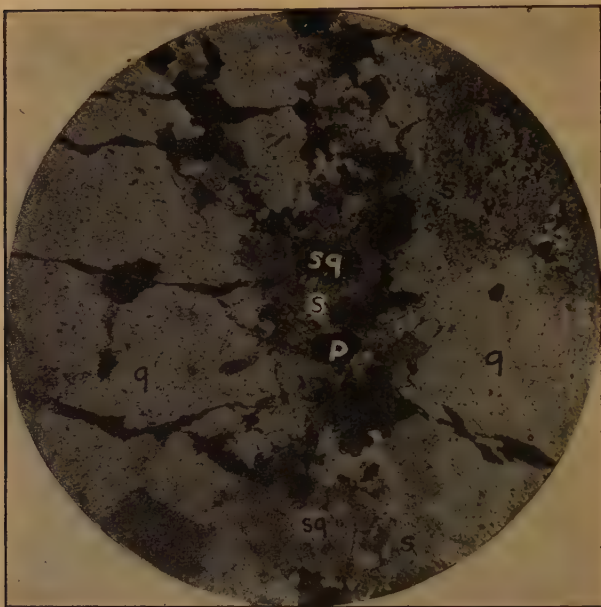


FIG. 24.—VEINLET OF SECONDARY QUARTZ (sq) AND PYRITE (p) IN PRIMARY QUARTZ (q) OF THE GRANITE. SHOWS REPLACEMENT. SERICITE (s) FROM ALTERATION OF FELDSPAR AND ALSO IN VEINLET. CROSSED NICOLS. $\times 50$.



FIG. 25.—ON THE ROAD FROM TYRONE TO SHAFT NO. 2, SHOWING GOSSAN OUTCROPS.

which were developed after the principal period of mineral formation. The latter is sometimes well shown in the hand specimen, as in Fig. 20.

Chloritization, sericitization, and silicification represent the action of hydrothermal solutions. The zonal arrangement of the first two corresponds in a general way with the alteration at Butte, as described by Kirk.¹⁷

Kaolinization.—Meteoric waters have developed near the surface large amounts of kaolinite which give the rock a soft, chalky appearance. As with the secondary metallic minerals, kaolinite extends to varying depths, depending upon the permeability of the rocks.

6. *Genesis of the Ores*

The location of the Burro Mountains deposit on the contact of a post-Cretaceous intrusion of monzonite, its formation after that intrusion and yet previous to a Tertiary volcanic breccia, and a mineralizing activity which is generally considered to-day to be due to hot solutions, point to the monzonite as the prime factor in its existence. In this respect it corresponds with the many deposits of the southwestern part of the United States which are closely connected in origin with monzonite intrusions.

A fractured zone helped to determine the location of the deposit by furnishing channels for the easy circulation of the hot ascending solutions that were given off by the congealing monzonite magma. These hot solutions chloritized the ferro-magnesian minerals and sericitized the feldspars of the country rocks; they deposited quartz and pyrite in the walls of the fissures wherever minor fractures would allow them to penetrate; and they filled the fractures themselves with an aggregate of quartz, pyrite, chalcopyrite, and sphalerite, probably because of a decrease in pressure and temperature as the solutions rose. Chalcopyrite was scarce in the original aggregate, and hence copper very low, but since the fractures were first exposed at the surface by erosion, meteoric waters have been effecting a concentration. This has been brought about by oxidation of the copper to soluble salts, downward percolation, and precipitation as chalcocite upon reaching the reducing environment of the underground water.

Both primary mineralization and downward enrichment were given the most favorable opportunity for their activity where the fracturing was most open, a condition which seems characteristic of the central part of the fractured zone, and hence the best orebodies are found in that portion.

¹⁷ Kirk, C. T.: Conditions of Mineralization in the Copper Veins at Butte, Mont., *Economic Geology*, vol. vii, No. 1, p. 35 (Jan., 1912).

The district is still comparatively undeveloped. Many new orebodies will undoubtedly be disclosed as mining is carried on, and the future of the district seems bright.

DISCUSSION

A. C. LAWSON, Berkeley, Cal.—One point to which Mr. Graton (in summarizing the paper for us) called attention is that the author had not explained why some copper was fixed in the zone of oxidation. I recollect very well puzzling over that question at Ely, Nev., some years ago. Mr. Sales, in his paper on Butte,¹ suggests the explanation. It is due according to him to the paucity of pyrite in the primary ore. Where there is abundant pyrite sufficient sulphuric acid is formed by oxidation to take all the copper into solution and carry it to lower levels; but where the pyrite is scant so that insufficient sulphuric acid is formed, some of the copper is carbonated and so fixed in the gossan. The hypothesis, or some modification of it, as for example the combination of the sulphuric acid with lime, would appear to explain satisfactorily the fixation of the copper in the gossan as carbonate. I have, however, had no occasion in recent years to study the phenomenon.

L. C. GRATON, Cambridge, Mass.—As I remember, the rocks in which the chrysocolla occurs in the Burro Mountains are both silicified and kaolinized.

It seems probable that in surroundings deficient in carbon dioxide, complete oxidation of chalcocite may yield chrysocolla up to a maximum of half the copper in the chalcocite; while in rocks, like garnet contact zones and certain easily decomposable porphyries, surface decomposition appears often to yield silica in the necessary form to convert the copper of primary sulphides into chrysocolla immediately and directly upon oxidation.

¹ Reno H. Sales: Ore Deposits at Butte, Mont., *Trans.*, xlv, p. 49 (1913).

The Copper Deposits of San Cristobal, Santo Domingo

BY THOMAS F. DONNELLY, E. M., NEW YORK, N. Y.

(San Francisco Meeting, September, 1915)

Introduction

THE Province of San Cristobal is situated on the south side of the island of Santo Domingo about 25 miles west of Santo Domingo city, the capital of the republic. The copper mineralization is found about



FIG. 1 —CAMP BUCARO. THE TOP OF THE RIDGE IN THE BACKGROUND MARKS THE CONTACT OF THE LIMESTONE AND TUFFS.

8 miles north of the town of San Cristobal, in the section known as Bucaro Hill; in San Francisco Hill; on the Nigua River; and on the Jaina River. The district is reached by automobile from Santo Domingo city over an excellent road for 25 miles to San Cristobal, and then, by horseback 6 or 8 miles up the Nigua River to what is locally known as "Camp

Bucaro" (Fig. 1). The trip from the capital takes about 3 hr. The camp is really not more than 10 miles from the sea; yet, since the coast offers no suitable harbor in this vicinity, Santo Domingo city is its only available port.

The district under consideration is situated in what might be called the foothills of the southern watershed of the main Cordilleras, a range



FIG. 2.—LOOKING UP THE NIGUA RIVER AT CAMP BUCARO.

which traverses the center of the island in a slightly north of west direction. The mountainous nature of this section of the country extends almost to the sea, so that at Camp Bucaro the river bed has an elevation of 500 ft. above the sea level. In this vicinity the high peaks rise from 1,200 to 1,500 ft. Up the Nigua River, the mountains become higher and more rugged until, 40 or 50 miles up, where the main ridges are encountered, they attain an elevation of 7,000 ft., with some peaks over 10,000 ft. above tide.

The Nigua Valley is narrow, steep, and rugged. The river itself is small—hardly more than a creek—and shallow enough to be crossed almost anywhere by stepping from stone to stone. However, it has a steep grade, and during the rainy season is often impassable (Fig. 2).

The climate is semi-tropical and very healthy. The thermometer registers around 65° F. during the evenings and night, and rises to about 85° F. at mid-day. There is no regular rainy season in this section. The most rain falls between June and October; but there is rain throughout the year.

General Geology

Briefly speaking, the region consists of a Cretaceous limestone in contact with post-Cretaceous igneous rocks, which are cut in many places by trachyte dikes. These igneous rocks are principally basaltic and basic tuffs¹ and volcanic ashes.² The general direction of the contact is approximately northwest-southeast. The country has suffered violent fracturing and faulting during at least two separate periods. The older series has a general strike of about east and west, and the later system strikes on an average of about N. 30° E. This later system has been productive of the greater movement. Mineralization has taken place in both series of fractures, and each has had its influence on the ore deposition of the district.

The country presents an intricate network of fractures, extending in every direction. On the hills the surface has been greatly weathered, disintegrated, and leached; whereas in the river bed the tuffs are exposed in hard, compact, fresh form, indicating that the atmospheric agencies have exerted a greater influence than the river waters (Fig. 3).

Economic Geology

There are two promising sources of ore in the district: (1) the deposits that should occur at or near the contact of the igneous rocks and the limestone; and (2) the fissure veins.

Of the former the most attractive prospects are found in the north-western quarter of the Bucaro claim on Bucaro Hill (Fig. 4). In this section the following interesting conditions exist:

¹ There seems to be some confusion as to the exact nature of tuffs. They are the fine, fragmental ejections from the explosive eruptions of a volcano. They may afterward be water sorted or cemented to a firm rock. Coarser *ejectamenta* are called volcanic breccias, but in neither do we see much sorting unless produced by subsequent erosion. Tufa is sometimes used in this sense, but the practice is wrong, and should be discouraged. A tufa is a cellular deposit of a mineral spring. It is usually calcareous or siliceous. The term should be confined to this material and not be confused with tuff.

² *Mining and Scientific Press*, vol. xcv, No. 10, p. 305 (Sept. 7, 1907).

1. There is a contact of Cretaceous limestone and igneous rocks—principally tuffs and volcanic ashes—the course of which is approximately northwest-southeast. The limestone is tilted nearly vertical and strikes almost north and south.

2. In the igneous rocks there are numerous stringers of ore and a couple of veins that give promise of productiveness. The general strike of this mineralization is about east and west, and hence will ultimately meet the contact.

3. There is a later system of faults which strike approximately north,



FIG. 3.—UNALTERED TUFF IN THE BED OF THE NIGUA RIVER, SHOWING FRACTURE PLANES IN ALL DIRECTIONS.

and south and dip to the west. This system displaces the former system, and toward the western end of the property seems to have changed the dip of the older series of fractures—near the surface at least—from north to south, or toward the limestone. This later series evidently carries no original copper mineral, and the fissures are generally barren except for some “drag” where they cut the earlier fractures. There are, however, north and south dikes that carry what seems to be original disseminated pyrite, but no copper minerals.

The vein filling consists of quartz, limonite, some specularite, hematite, malachite, chalcopyrite, bornite, and occasionally chalcocite and cuprite. The ore deposition probably took place first in the original



FIG. 4.—BUCARO HILL, FROM SAN FRANCISCO HILL.



FIG. 5.—OUTCROP ON SAN FRANCISCO HILL, SHOWING SHATTERED CONDITION OF COUNTRY ROCK.

fissures, as suggested below; but subsequent disturbances have so shattered the country in general, and the minerals have been leached and re-deposited in the walls to such an extent, that it is now impossible to determine the original walls. This condition may be confined to the surface only, which has been greatly weathered and leached; and the veins may be better defined in depth.

It seems to me that the conditions would be ideal for the formation of deposits along the contact of the limestone and the tuffs in the vicinity of where this series of mineralized fractures meet it—a zone of



FIG. 6.—SAN FRANCISCO HILL, SHOWING OPENING ALONG CROWN COPPER VEIN. AT THE TOP OF THE HILL THE VEIN IS 30 FT. WIDE.

about 4,000 ft. in length. The present development has for its object the exploration of this zone.

The fissure veins are best exemplified on the north side of the Nigua River, in the vicinity of San Francisco Hill. Here they are quite different in appearance from those around Bucaro. The walls are better defined and quartz is much more prominent in the vein filling.

The older fissures on San Francisco Hill strike a little north of east (although this varies somewhat) and dip to the north (Fig. 5). The veins range from fine stringers to a width of 3 or 4 ft. This system of veins has been displaced and offset by a later period of faulting,

with a result that seldom more than 100 or 200 ft. of vein are found in one continuous line. The vein filling is white sugary quartz, carrying as its principal mineral bornite, with some minor quantities of azurite and malachite. The precious-metal values are fairly good, but not uniform.

The more promising veins are the later ones, or those that show a strike of about N. 30° E. The dip of these varies considerably, those on

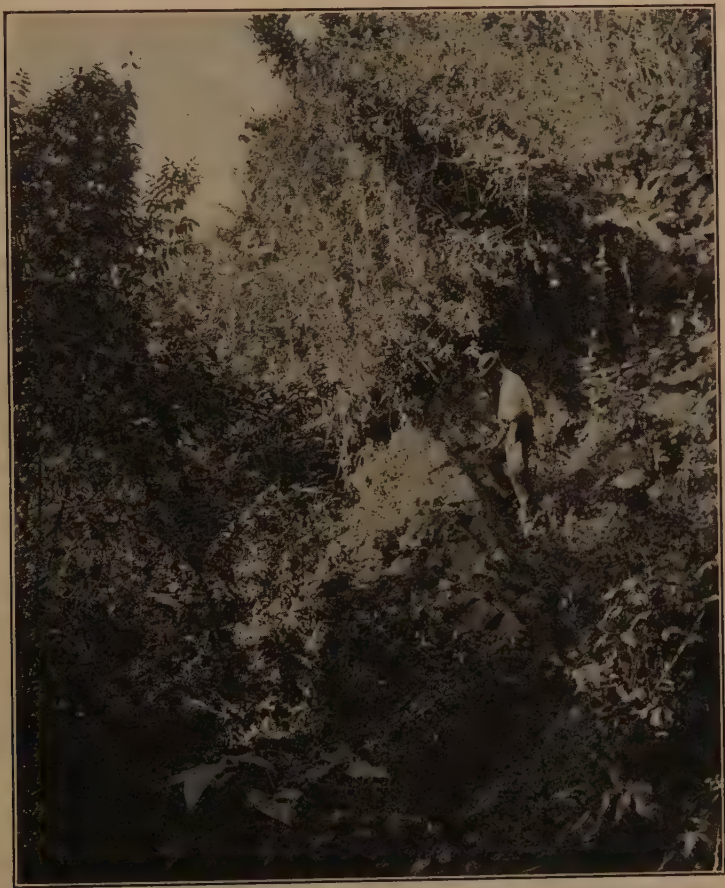


FIG. 7.—PLATINITA CREEK. OUTCROP NEAR HEAD.

the eastern portion of the property dipping generally to the east, and those on the western end to the west. The vein on which most work has been expended in this region is locally known as the Crown copper vein (Fig. 6). It has been exposed across San Francisco Hill by pits, shafts, and stripping for a distance of about 4,000 ft. This fissure measures in one place 30 ft.; at the Nigua River it is 10 ft. wide. I do not think, however, that it will average nearly this width. The vein appears to have experienced, after its original mineralization, a pronounced fault move-

ment, to which its peculiar character is due. In the hanging wall is found, in places, a layer of rich bornite ore a few inches thick; and filling the rest of the fissure is a talcose gouge, which shows by the presence of rounded nodules of quartz and chalcopyrite covered with slickensides that a great amount of movement has taken place. It is still a question, whether this last-mentioned mineralization took place originally in the fissure, or is a by-product from the older veins, dragged in by the movement. Work is now in progress to determine this.

About the vicinity of Platinita Creek, there are outcrops of rich ore (Fig. 7) with a general strike of N. 30° E. showing indications of strong veins. Hardly any work has been expended on these hitherto but plans have been made for their early exploration.

Genesis of the Deposits

The presence of copper in tuffs is not uncommon. De Launay has observed that the andesitic tuffs at the Boleo mines, near Santa Rosalia, Lower California, have been the probable source of copper for these deposits.³ He concludes that the deposits are due to the sedimentation aided by the vein-forming waters that emerged during the tectonic movement, and that the gold, pyrite, copper, etc., were precipitated and deposited simultaneously with the pebbles of the conglomerate.³ Weed takes exception to this view, and believes that the evidence would indicate that the circulation of mineralizing waters after the sedimentation was the agent of deposition.⁴ Ransome says that in many of the mines northwest of Globe, Ariz., as the Black Warrior, Black Copper, and Geneva mines, the dacite tuffs are richly impregnated with chrysocolla.⁵ J. A. Dresser claims that the chalcopyrite lenses in Quebec are in sheared volcanic rocks that were subsequently changed to schists. He states that during the expulsion of these volcanic rocks copper, gold, silver, etc., were also emitted, to be afterward concentrated by metamorphism and meteoric waters.⁶ The last word on the Rio Tinto deposits in Spain places them also in sheared eruptive rocks that were subsequently changed.⁷

In the above instances the deposits occur either in lenses or veins along more or less defined bedding planes. In the San Cristobal district, the ore is found in fissures that run in all directions.

In the San Cristobal district there are good reasons to believe that

³ De Launay: *Les Richesses Minérales de l'Afrique*. Translation from the French, *Engineering and Mining Journal*, vol. lxxv, No. 14, p. 519 (Apr. 4, 1903).

⁴ *Copper Mines of the World*, p. 245 (1907).

⁵ *Professional Paper No. 12*, U. S. Geological Survey, pp. 155 to 159 (1903).

⁶ *Journal of the Canadian Mining Institute*, vol. v, pp. 81 to 86 (1902); *Engineering and Mining Journal*, vol. lxxiii, No. 12, p. 412 (Mar. 22, 1902).

⁷ De Launay: *Traité de Métallogénie* (1913).

the copper, in part at least, has been derived from the tuffs. On the surface where these igneous rocks are severely weathered and disintegrated, very little or no copper is present; but in the tunnels in Bucaro Hill, it is quite evident that the copper is disseminated through the unaltered tuffs. A slight green stain of copper carbonate may be seen almost anywhere.

The limestone escarpment that runs along the top of Bucaro Hill and denotes the probable contact of the limestone and the tuffs is, in all likelihood, the result of a normal fault. Probably the same forces and movement that caused this produced simultaneously the series of east and west fractures and fissures. This seems to be indicated by the fact that the fissures with this general direction are more numerous and pronounced near the contact, where the forces have been most potent. The dying phase of this activity probably marks the beginning of the vein formation. The atmospheric waters, percolating through the copper-bearing tuffs, met the ascending magmatic waters; and these, rising together in the fissures, deposited their mineral contents in them. The later, or north and south, system of faults, further shattered the district, made the country more pervious to the action of the meteoric waters, and assisted in the lateral segregation. The trachyte dikes that have the same general strike may have come up at this time, and the emanations from these may have been prolific of further mineral deposition. These dikes carry considerable pyrite, but I was unable to find any copper in them in determinable quantities.

Conclusions

One is greatly handicapped in the study of the country: first, because of the dense tropical growth; second, because of the weathering, which has caused a great deal of disintegration and leaching at the surface; and third, on account of the lack of development work, especially underground. I spent a little over four months in the country, and most of this time was devoted to the surveying and mapping of the district preliminary to a more detailed study. I feel, therefore, that the knowledge thus gained is of a superficial nature, but hope that it may be of benefit to subsequent investigators.

I was very favorably impressed by the district. Although still in the "prospect" stage, the mineralization is so abundant and general that great hopes are entertained for the future if operations are properly directed. It will require money to explore and develop the country, and this money will have to be intelligently expended. A large sum has been spent by the various companies operating here, but only a small percentage of it has been used in real exploration work. This has had a tendency to destroy the confidence of investors; but the district itself is not at fault.⁸

⁸ *Engineering and Mining Journal*, vol. xcix, No. 15, p. 641 (Apr. 10, 1915).

Labor conditions are highly favorable. Native labor is employed, at 50 c. per day. Contracts are let to natives at an average of \$1.50 per running foot for tunnels 5 by 6 ft. in size. Of course the company furnishes all supplies. Native skilled labor can be had at from \$1.50 to \$2 per day. The natives for the most part are pure negroes, and, like all of that race, are great imitators, and, as a consequence, quick to learn. They are a simple, industrious, child-like people, very little addicted to the use of alcohol, and hence easily handled. There are now but few really good miners among them. They would, however, develop skill under some one who could teach them. Like all Southern laborers, they are inclined to be lazy when on day's pay but they work 12 or 15 hr. a day when on contract.

There is an abundance of water handy for mill operations and there is a good site for a power plant on the Jaina River some 40 miles away.

The hillsides are fairly well forested with hard woods. Mine timber, however, is very scarce.

The transportation problem is not as bad as it would at first seem. There are 8 miles of bad road from Bucaro to San Cristobal. This is a crude road, following to a large extent the bed of the Nigua River. There are no bad grades on it, but it is impassable during the seasons when the river is up. The company now operating here has hauled a considerable amount of machinery over this road—some pieces being as heavy as 5 tons—without experiencing any great difficulty. The present mode of transportation by ox teams is, of course, very slow. But a little work would put the road in such condition that a gasoline traction truck could be used over it. From San Cristobal to Santo Domingo city the road is macadamized all the way. It is a government road called the "Carretera" and is always kept in excellent condition. The Clyde Steamship Co. has made a rate of \$4 per ton for carrying concentrates from Santo Domingo city to New York. This rate is, of course, exorbitant, in view of the fact that the rate for shipping ore from Cuba is only 90 c. per ton. A little competition in the shipping field, and the active operation of the district, will do much toward lowering this rate to within reasonable bounds.

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DISCUSSION

F. LYNWOOD GARRISON, Philadelphia, Pa. (communication to the Secretary*).—The paper is of value in giving the public some first-hand information regarding a copper property that from various causes has attracted considerable attention, especially in Philadelphia, and which has also been the source of several fantastic flotations that had little to recommend them.

Since 1906, no work has been done on this property under my direction, with the exception of a topographic survey made last winter by Mr. Donnelly, which was intended as the first step in a sound and logical system of development. From 1906 to 1915, desultory attempts were made to develop these orebodies, but as the work was not in competent hands much money was wasted and even more disappeared in ways regarding which it is perhaps not best to inquire. The result is, this interesting property has now practically no more ore developed upon it than in 1906, despite the fact that a new 100-ton mill is being erected, although, as I have intimated, it is today not much more than a prospect.

Turning now to the scientific and more agreeable side of the subject, it is interesting to note that these Santo Domingo copper ores are invariably found in volcanic tuffs and that, in a geologic sense, they are all very old, for there are no evidences of recent volcanic activity in the greater Antilles, of which this island is the second largest. The existence of copper in tuffs in different parts of the world is a notable fact. Often, if not usually, it is present in mere traces, but not infrequently we find it in sufficient amount to be of economic importance. Thus, for example, chalcopyrite and bornite occur in the Tertiary tuffs of Japan, and a number of producing copper mines of that country are either actually in, or closely associated with, rocks of this character. Whether the copper minerals are original primary ores in the tuffs, or were derived from accompanying eruptive dikes, appears not yet to have been determined in any instance. In most of the Japanese cases of which I can find any record the tuffs appear to be associated with shales and limestones. These Japanese copper ores also carry some gold, especially in the upper parts of the veins; in this they resemble the Santo Domingo

* Received June 29, 1915.

deposits which not infrequently carry as much as \$6 or \$7 per ton of gold on the outcrop. As a rule the content of gold in both the Japanese and Santo Domingo deposits decreases in depth very rapidly, and a few feet down the average tenor appears to be about \$1.50 per ton. The explanation is simple; the copper minerals being more soluble they are leached out on or near the surface leaving the gold behind. It is not, however, so easy to account for the fact that visible gold is often present in the outcrop oxidized ores and never perceptible in the sulphides. The only explanation that offers itself for this peculiarity is that the gold may in some manner be dissolved and then reprecipitated by the ferrous sulphate certain to be present in waters flowing through mineralized and jungle-covered rocks.

The Braden Copper mines in Chile are reported to be in the perimeter of a volcanic vent filled with tuffs. The copper sulphides and metallic copper occur in seams and veinlets in diorite and brecciated contact material.¹

It is a notable fact that of late years volcanic tuffs are becoming more and more to be recognized as the habitat of metallic minerals and ores. Rocks of this character are not always readily recognized as such, being often denominated andesite, olivine-basalt, diabase, or the general term basic eruptives. In view of the fact that a real tuff reaches the surface of the earth as a volcanic ejectment in the form of dust (sometimes termed ash or mud) to be consolidated afterward by surface phenomena, whereas effusive rocks such as andesite, diabase, diorite, etc., appear as magmas or lavas that have flowed out from the interior of the earth, this confusion of terms is unfortunate and bespeaks a carelessness of observation and diagnosis. It is evident on the whole that copper minerals have an affinity for tuffs which must be recognized, and it may not be going too far to assume that such volcanic dust is the original source of the copper we find in the fissures of the rocks which have resulted from the consolidation of these ejectments.

¹ *Engineering and Mining Journal*, vol. lxxxiv, p. 1060 (Dec. 7, 1907.)

The Formation of the Oxidized Ores of Zinc from the Sulphide

BY YINCHANG TSENSHAN WANG, NEW YORK, N. Y.

(San Francisco Meeting, September, 1915)

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I. INTRODUCTION

1. *Subject and Scope*

The formation of the oxidized ores of zinc has been recognized as a subject of great importance in economic geology. For many years these ores have been much sought, partly because they are more desirable for metallurgical treatment and partly because they are apt to be freer from gangue than is the sulphide of zinc. Up to very recent years the oxidized ores were the principal source of the metal and at present they still furnish a large portion of the commercial production.

The oxidized ores of zinc include six minerals, as shown in the following table:

Mineralogical Name	Chemical Name	Chemical Composition	Constituents, Per Cent.			
			Zn	SiO ₂	Fe	Mn
Calamine....	Basic silicate of zinc	2ZnO.SiO ₂ .H ₂ O or Zn-(OH) ₂ .ZnSiO ₃	54.20	25.0
Smithsonite.	Zinc carbonate.....	ZnCO ₃	52.00
Hydrozincite	Hydro-carbonate of zinc.....	ZnCO ₃ .2Zn(OH) ₂ ...	60.0
Willemite...	Anhydrous silicate of zinc.....	2ZnO.SiO ₂ or ZnO.ZnSiO ₃	58.50	27.0
Zincite.....	Zinc oxide.....	ZnO.....	80.30
Franklinite..	Iron-manganese-zinc oxide.....	(FeZnMn)O(Fe-Mn) ₂ O ₃	12.00	42.0	15.0

Only the first three, calamine, smithsonite and hydrozincite have been considered in this investigation. Wherever the terms "oxidized ores" and "oxidation products" are used in this paper they are, unless otherwise indicated, strictly limited to these three ore-minerals of zinc.

These oxidation products are universally believed to be derived from the sulphide of zinc, as their original mineral. The chemical reactions of the change, however, are not fully understood and have not, heretofore, been studied experimentally in the light of physico-chemical principles. The influence of oxidation by surface waters and of other metallic compounds which are associated with the blende, is only inferred from field relations. At the suggestion of Dr. J. F. Kemp, the writer undertook a series of experiments with the hope of clearing up some important points in connection with the chemistry of the precipitation of the three oxidized ores and in particular, of establishing the peculiar set of conditions under which the antecedent oxidation of the blende has taken place.

The principal part of the experiments is, in accordance with the geological evidence, limited to wet methods, and deals with a single substance, or a group of substances in solution, which are capable of effect-

ing the solution of the blende or of converting its oxidation product into a carbonate or a basic silicate, as the case may be.

2. Acknowledgments

The writer takes great pleasure in acknowledging his indebtedness to his professors and friends for aid received in the preparation of this paper. First among them is Professor J. F. Kemp, of Columbia University, under whom this work was done. Throughout the entire laboratory work, and especially on the experimental side, he has manifested great interest in the results as they appeared. Whatever merits this paper may have are largely due to his valuable assistance and advice. For many useful criticisms relative to the petrographic feature of the work, the writer is indebted to Professor C. P. Berkey, of the same University. The writer's thanks are also due to Professor T. C. Chamberlin, of the University of Chicago, for inspiration and encouragement through correspondence and otherwise during the execution of this research, and to Dr. R. T. Chamberlin for many helpful suggestions. To Professor R. J. Colony of Cooper Union of New York City, the writer is indebted for checking the results of optical examinations of the crushed fragments. Acknowledgments to other friends have been indicated in the course of the following pages.

3. Review of Previous Work

Field Data.—Field data concerning the paragenesis of the oxidized ores of zinc are abundant. The most suggestive contribution to the literature of the subject is by Dr. R. Beck.¹ In the discussion of the zinc deposits at Raibl in Carinthia, Austria, he regards the oxidized ores of zinc in that region as the results of extensive metasomatism, and says that the limestone, the inclosing rock of the ores, "is directly replaced by calamine and the structure of the original rock is retained by the calamine and carbonate of zinc." Our general experience shows that calamine and smithsonite are derived from blende. Therefore, the solutions which replaced the soluble limestone presumably obtained their salts by the oxidation of sphalerite and the process was essentially one of secondary enrichment. That is to say, the oxidation and solution of the sphalerite must equally depend: (1) on the action of the oxidation products of pyrite or marcasite on the sphalerite in presence of water (these iron sulphides in nature are invariably associated with zinc sulphide); (2) on the direct oxidation of sphalerite as a result of electrolytic action²

¹ R. Beck. *Lehre von den Erzlagerstätten*, Band 2, p. 596 (1901).

² V. H. Gottschalk, and H. A. Buehler. Oxidation of Sulphides, *Economic Geology*, vol. vii, p. 28 (1912).

TABLE I.—*Analyses of Mine Waters and Well Waters in Parts per Million*

	1	2	3	4	5	6	7	8	9	10
S*.....	184.2	1886.0	2033.4	3200.3	—	221.4	429.0	154.3	29.7	161.0
Cl.....	0.0	0.0	0.0	0.0	0.0	0.0	12.8	67.45	3.5	—
Free										
H ₂ SO ₄ ...	0.0	220.5	39.0	1568.0	333.2	0.0	0.0	0.0	0.0	0.0
Combined										
H ₂ SO ₄ ...	0.0	0.0	6060.9	8032.9	0.0	0.0	0.0	0.0	0.0	0.0
Metallic										
Fe.....	0.0	670.0	362.0	523.5	465.0	—	0.0	—	1.0	15.5
Fe ⁺⁺	0.0	490.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	—
Fe ⁺⁺⁺	0.0	180.0	293.0	1091.5	—	0.0	0.0	3.0	—	—
CaO.....	354.1	—	700.5	—	—	—	501.0	333.9	—	—
MgO.....	31.4	—	0.0	—	—	—	—?	108.6	—	—
Pb.....	0.0	0.0	0.30	0.0	0.0	0.0	0.0	0.0	1.3	0.0
Metallic										
Zn.....	20.0	1432.5	80.50	2250.0	910.0	20.9	257.0	1.7	7.3	70.0
Acidity						Alkali ne				

	11	12	13		14	15	16	17
KCl.....	9.5	6.1	0.0	SO ₄	233.4	222.4	215.9	104.0
K ₂ SO ₄	0.0	0.0	0.0	SiO ₂	6.8	8.0	2.1	18.0
NaCl.....	0.0	1.3	4.5	Al ₂ O ₃	2.8	0.7	2.8	0.0
Na ₂ SO ₄	130.3	38.5	30.8	FeO&Fe ₂ O ₃	6.7	9.0	6.8	9.5
Al ₂ (SO ₄) ₃ ..	68.3	73.6	3.3	MnO.....	2.7	5.9	8.3	0.0
CaSO ₄	1528.0	924.6	371.5	ZnO.....	23.5	5.9	3.5	11.1
Ca(HCO ₃) ₂	0.0	0.0	0.0	Cl.....	trace	trace	trace	12.4
MgSO ₄	trace	192.6	60.4	CaO.....	148.5	101.4	127.0	65.0
FeSO ₄	611.9	387.7	2.6	MgO.....	28.4	24.7	21.7	24.9
PbSO ₄	0.04	0.0		Insoluble ..				
ZnSO ₄	586.2	851.7		residue....	17.5	15.8	21.0	0.0
SiO ₂	23.2	23.2	7.4
H ₂ SO ₄	170.9	251.7	0.0
CO ₂ (free)..	12.0	87.0	19.7

1. Oronogo Circle mine, Oronogo, Mo.
 2. Duenweg mine, Joplin district, Mo.
 3. Continental mine, Joplin district, Mo.
 4. Mary mine, Joplin district, Mo.
 5. Diamond Jack mine, Carterville, Mo.
 6. 9th Shaft, Oronogo Circle mine, Oronogo, Mo.
 7. Well on Centre Creek.
 8. Well on Main Street, Joplin, Mo.
 9. Centre Creek water.
 10. Centre Creek water.
 11. Arkansas mine, Joplin district, Mo.
 12. Victor mine, Joplin district, Mo.
 13. O. K. mine, Joplin district, Mo.
 - 14-17. Mine waters of Rothschoenberger Stollen, near Freiberg, Germany.
- = No determination was made of the element.
O = The element was absent.

* The sulphur in all cases exists as sulphate.

due to the contact of sphalerite and iron sulphides; or (3) on both of these actions.

In addition to this statement of Beck's, we have the evidence afforded by analyses of mine waters and of well waters from zinc districts.

"While the waters taken from a mine may differ slightly in composition from the ground waters that seeped through the undisturbed deposit, in that they are probably more dilute on account of the freer circulation of the solutions, and are also more highly oxidized on account of greater access to the atmosphere, yet nevertheless their composition approaches fairly closely that of the original ground-waters."³

The Table I comprises typical analyses⁴ of the waters from zinc mines both in Europe and in America, and of the waters from wells in the zinc district of Joplin, Mo. The figures given are all in parts per million.

On inspection of these data, it will be noted that sulphur in all cases exists as sulphate, indicating as a distinctive feature of these waters, the predominance of the sulphates of zinc, iron, calcium and generally magnesium, over other salts. It has also been noted, according to Mr. Waring,⁵ the Chemist of Webb City, Mo., that the iron in the waters freshly drawn from the depths of the Joplin mines, is always present as ferrous sulphate, while the waters from the surface levels frequently contain ferric sulphate. Further evidence has recently been shown by Hodge⁶ as to the presence of ferrous salts in the waters at some depth and of ferric near the surface.

According to Allen⁷ and Wells,⁸ the excess of sulphuric acid formed by the oxidation of the iron sulphides is responsible for the increased solubility of the metallic sulphides in association. This assumption does not seem to be entirely consistent with the experimental facts,⁹ or to be generally supported by chemical analysis. Mr. Waring,¹⁰ speaking from his wide experience in the zinc mines of Joplin district, Mo., says that free sulphuric is not always present; in some cases, the waters that are quite rich in zinc sulphate contain no sulphuric acid. On the contrary they gave a strong alkaline reaction to methyl orange, frequently containing a considerable amount of calcium bicarbonate. The presence of the latter compound in solution may be a factor in the process of the

³ N. C. Cooke. *Journal of Geology*, vol. xxi, p. 5 (1913).

⁴ For the analyses Nos. 1 to 10, the writer is indebted to W. G. Waring of Webb City, Mo., who has very kindly placed at his disposal the data which for the first time appear in print. For Nos. 11 to 13 inclusive, the writer has quoted from a paper by Messrs Buehler and Gottschalk, *Economic Geology*, vol. v, p. 34 (1910); Nos. 14 to 17 from *Lehre von den Erzlagertstätten*, R. Beck, vol. ii, p. 402 (1901).

⁵ Private communication.

⁶ E. T. Hodge. *Economic Geology*, vol. x, p. 123 (1915).

⁷ E. T. Allen. *Economic Geology*, vol. v, p. 387 (1910).

⁸ R. C. Wells. *Economic Geology*, vol. v, p. 480 (1910).

⁹ Gottschalk and Buehler. *Economic Geology*, vol. vii, p. 27 (1912).

¹⁰ Private communication.

formation of zinc carbonate and will be discussed experimentally later in this paper.

Experimental Data.—In the literature of this subject no experimental data bearing on this problem were found. Those who have dealt with the question in the laboratory have rarely worked under the conditions which commonly obtain in nature; *i.e.*, low temperatures, low pressures, small concentrations, and the combination of all these conditions, presenting a narrow range of possibilities for imitative synthetic studies. H. de Senarmont¹¹ showed that zinc carbonate can be formed from zinc chloride and calcium carbonate at 150° C. Gorgeu¹² in 1887 claimed that he obtained the anhydrous silicate and the oxide of zinc (willemite and zincite) by fusing a mixture of chemical compounds containing the required constituents. His experiments have been repeated by the writer, but without producing the same results.¹³ No attempt has heretofore been made, even using high temperature, to produce synthetically the basic silicate of zinc (calamine).

The investigations of Allen and Creushaw¹⁴ on zinc compounds have no bearing on this problem, since their work was confined to the study of zinc sulphides.

4. Plan of Present Investigation

The present work was divided into two parts, experimental work and microscopic studies of thin sections. Experimentally the main purpose was to find: (a) what is the exact nature of the compound or compounds, commonly associated with the sulphide of zinc, which cause it to be oxidized and carried into solution in meteoric waters; (b) what are the most common reagents or compounds which are likely to have caused the precipitation of carbonates or the basic silicate of zinc; (c) what are the peculiar conditions under which these oxidation products may be obtained in the laboratory; (d) to what extent the formation of these oxidation products is limited by ordinary laboratory conditions; and (e) what influence is exerted by temperature. The microscopic study was done chiefly in the hope of obtaining further evidence throwing more light on the problem.

The experimental work here described deals with the following questions:

1. The solvent effect of iron sulphates on sphalerite.
2. The solubility of sphalerite in a salt solution of silver, of copper and of lead.

¹¹ Doelter y Asterich. *Handbuch der Mineralchemie*, Band 1, p. 499.

¹² Bulletin 10, *Société Française de Mineralogie*, pp. 30 to 39.

¹³ See the discussion under "The Influence of Temperature on the Formation of Calamine," Experiment 12.

¹⁴ *American Journal of Science*, 4th series, vol. xxxiv, pp. 341 to 360 (1912).

3. The reactions of carbonates and bicarbonates with solutions of zinc salts.

4. The reaction of zinc sulphate solution with pure and impure limestones.

5. The effect of carbon dioxide upon the basic silicate and the carbonates of zinc in water, and its bearing upon the conditions of their precipitation.

6. The reactions of a mixture of hydroxide and silicate of sodium with dilute solutions of a zinc salt.

7. The influence of temperature on the formation of calamine, and its possible effects on natural calamine and smithsonite.

On account of the wide range of the subjects involved in the experiments, each series will be described and discussed under its own heading. Unless otherwise indicated, all of these experiments were conducted under ordinary conditions of temperature and pressure. Concentrations of solutions were small, deci-normal in some cases, and fiftieth in others; these concentrations may slightly differ in behavior from those commonly obtained from the ground waters, but the difference is only in the speed, and not in the nature, of the chemical reactions.

The paper also includes a discussion, based on the suggestion of Dr. Kemp¹⁵ as to the origin of the zinc deposits at Franklin Furnace, and Ogdensburg, N. J., of the effect of heat on calamine and smithsonite, with possible changes to willemite and zincite respectively.

Method of General Procedure.—Unless otherwise indicated, the mineral to be tested was first powdered; and only the portion which passed 50- or 100-mesh sieve, as the case might be, was used. Chemically made zinc compounds were also used and are indicated in each case. The amounts of reagents in solution, as to their constituents, were definitely known whenever it was deemed necessary. In all cases, the precipitate, the residue if any, and the liquid contents in filtrates, were determined quantitatively. The comments which usually follow the record of the experiment were made in each case in consequence of the laboratory results. In determining the temperatures of formation of the oxidation products (here including willemite and zincite but excluding hydrozincite) both the quantitative analysis and microscopic examination of the crushed fragments were employed.

II. EXPERIMENTAL WORK

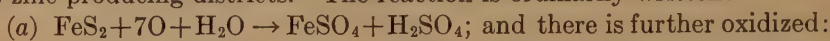
A. The Solution of Sphalerite

1. *The Solvent Effect of Iron Sulphates upon Sphalerite.*—As already remarked, there is commonly present in the freshly drawn water from the depths of zinc mines, such as those of Joplin, an iron salt in solution,

¹⁵ *Transactions of New York Academy of Sciences*, vol. xiii, p. 92.

which, if drawn from some depth, is invariably in the form of ferrous sulphate, and doubtless indicates the oxidation of the blende to some extent at the expense of the reduction of ferric sulphate.

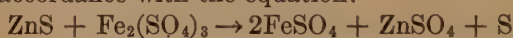
The ferric sulphate is presumably produced by the oxidation of an iron bisulphide (pyrite or marcasite) which occurs not infrequently in the zinc-producing districts. The reaction is ordinarily written:



(b) $2\text{FeSO}_4 + \text{H}_2\text{SO}_4 + \text{O} \rightarrow \text{Fe}_2(\text{SO}_4)_3 + \text{H}_2\text{O}$. This ferric solution may hydrolyze to



Reaction (c), however, occurs only in the absence of such minerals as blende. In blende-bearing strata the reaction (c) does not occur; at least, no such case has ever been reported. On the contrary, as shown by experimental results recorded below, when this strongly oxidizing agent is held in waters which percolate into blende-bearing rocks, it will react with the sphalerite and allow zinc to pass into solution. The reaction is in accordance with the equation:



In order to determine just what proportion of the ferric sulphate is necessary to oxidize one part of sphalerite, the following experiments were undertaken:

Experiment 1.—A piece of sphalerite obtained from Joplin, Mo.,

	I (a)		I (b)		II	III		IV	
	Actual amounts originally present in grams				Conditions in the filtrate before analysis. Grams.	Filtrate		Residue	
	In solution		In solid						
	Element	Compound	Element	Compound		Grams	Per Cent.	Grams	Per Cent.
S.....			1.598				0.3055	18.84*	
Impurity.....							0.0132		
Zn.....			3.262				trace		
Fe ⁺⁺					0.8794	6.304			
Fe ⁺⁺⁺	1.8636				0.9833	7.049			
SO ₄	4.8027				8.8270	63.270			
Zn.....					3.2550	23.330			
FeSO ₄					2.3900				
Fe ₂ (SO ₄) ₃		6.6663			3.5190				
ZnSO ₄					8.0350				
ZnS.....				4.860					
Total...	6.6663	6.6663	4.860	4.860	13.9447	13.9447	99.953	0.3187	18.84

*This shows that the rest of the sulphur (81.16 per cent.) has been oxidized into sulphate.

and known to contain 0.9972 g. ZnS for every gram of the sample was ground to pass a 50-mesh sieve; 4.873 g. of this material (on account of its impurity, this amount is a little less than $\frac{N}{10}$ of its molecular weight) was digested in 1,000 c.c. ferric sulphate solution, containing 6.663 g. of the salt (approximately $\frac{N}{10}$). The reaction is manifested as soon as the sphalerite is placed in the ferric solution. The mixture was allowed to stand in a 2-liter bottle, whose mouth was closed with a bunsen valve, and also attached by a side glass tube to an air tank from which ordinary air was allowed to pass slowly into the mixture 15 min. at a time, three times a day, for a period of four weeks. The foregoing table shows the analyses of both the residue, which weighed 0.3204 g. and the filtrate.

Experiment 2.—An equal amount, 4.873 g. of lump sphalerite from the same sample was treated in 1,000 c.c. $\frac{N}{10}$ ferric sulphate solution. In the same way as in Experiment 1, ordinary air was allowed to pass slowly into the mixture three times a day, 15 min. at a time, for a period of four weeks. The residue weighed 2.0154. g. The following analyses were obtained from the residue and the filtrate:

	I (a)		I (b)		II	III		IV	
	Actual amounts originally present in grams				Conditions in the filtrate before analysis. Grams	Filtrate		Residue	
	In solution		In solid						
	Element	Compound	Element	Compound		Grams	Per Cent.	Grams	Per Cent.
S.....			1.598					0.8572	53.23*
Impurity..								0.0136	
Zn.....			3.262					1.1490	
Fe ⁺⁺									
Fe ⁺⁺⁺	1.8636					1.1210	10.28		
SO ₄	4.8027					0.7413	6.795		
Zn.....						6.9380	63.59		
FeSO ₄					3.047	2.1120	19.36		
Fe ₂ (SO ₄) ₃ ..		6.6663			2.6523				
ZnSO ₄					5.213				
ZnS.....				4.860					
Total...	6.6663	6.6663	4.860	4.860	10.9123	10.9123	100.025	2.0198	53.23

* This shows that the rest of the sulphur (46.77 per cent.) has been oxidized into sulphate.

In both filtrates the iron and sulphate contents were determined with a known strength of K₂Cr₂O₇ and BaCl₂ solutions respectively. The

amount of the ferrous and ferric sulphates, together with that of zinc, in the filtrate was recalculated and recorded in column II. These analyses also show an excess of sulphur in the residue, indicating the presence of sulphuric acid, which is sometimes present with FeSO_4 and ZnSO_4 in the waters of zinc mines.¹⁶

According to the above results, there was practically no zinc remaining in the residue in experiment 1, and the amount of ferric (47.21 per cent.) reduced to ferrous, as a result of the oxidation of ZnS , in the first case is 12.96 per cent. less than in experiment 2, in which, in spite of the presence of more ferrous (60.17 per cent.) than ferric, only 65.34 per cent. of ZnS was oxidized. This difference is believed, other things being equal, to have been affected by the fine grinding of the ZnS in the first case, and its absence in the other.¹⁷

Whatever values are given ultimately for the rate of oxidation of zinc sulphide in the presence of ferric sulphate, the results of the above experiments show beyond any doubt that the chemical equation $\text{Fe}_2(\text{SO}_4)_3 + \text{ZnS} \rightarrow 2\text{FeSO}_4 + \text{ZnSO}_4 + \text{S}$ merely represents the chemical reaction and not the ratio. It is further shown that in contact with the oxygenated water of meteoric origin, which is no doubt an important factor in nature, even a comparatively small proportion of ferric sulphate will accelerate the rate of oxidation of the sulphide of zinc very materially. These apparent inconsistencies with the demands of the solution theory as required by the above chemical equation may be attributed to the influence of electrolytic action, as in the case with two different sulphides.¹⁸

The effect of ferrous sulphate upon ZnS is quite different from that of ferric sulphate. A solution of ferrous sulphate in contact with sphalerite produces no change whatsoever in either, if air is absolutely excluded. Different effects are observed, however, when the mixture is freely exposed to air. The following experiments were undertaken to demonstrate these points:

Experiment 3.—In this experiment the ferrous sulphate solution was obtained by allowing pyrites to oxidize under the influence of water and air and leaching the residue. The liquor was treated with scrap iron, and the neutral solution evaporated until a hydrate ($\text{FeSO}_4 \cdot 7\text{H}_2\text{O}$) was deposited. The salt is green in color. Two portions of it, containing approximately 3.799 g. FeSO_4 , without water, were treated in 500 c.c. of distilled water in two separate bottles (a) and (b), and each solution had a strength about $\frac{N}{10}$ of its molecular weight. The solution was then mixed with 4.873 $\frac{N}{10}$ g. of sphalerite, previously ground, and treated with 500 c.c. water.

¹⁶ See analyses 2, 3, 4, 5, in Table I.

¹⁷ *Economic Geology*, vol. v, p. 35 (1910).

¹⁸ *Economic Geology*, vol. vii, p. 28 (1912).

The mouth of the bottle (a), after mixing, was closed with its glass stopper, and the solution was left in the bottle undisturbed for four weeks. In (b) the water used was collected from rain and the mixture was allowed to stand in the bottle uncovered and freely exposed to air for the same period of time.

At the end of this period, each mixture was filtered separately. Both the residue and the filtrate in each case, were subjected to analysis.

In the analysis of (a) the filtrate was analyzed for Fe and Zn, but no Zn was found and only ferrous salts were present in solution indicating that no oxidation had occurred. The residue was still in the form of ZnS, as revealed by qualitative tests.

The analysis of (b) gave, from the residue,

	Grams	Grams	Grams
Fe ₂ O ₃	0.7343 = Fe	0.2569 = Limonite	= 0.9824
S	1.3671 = S	1.3671 = Sulphur	= 0.1554
Zn	2.4710 = Zn	2.4710 = ZnS	= 3.6827
Impurities	0.0137 = Imp.	0.0137 = Imp.	= 0.0137
H ₂ O	0.2251 = H ₂ O	0.2481*	
	<hr/> 4.8112		<hr/> 4.8342

and from the filtrate,

Fe	1.1410	= FeSO ₄	= 3.1030
SO ₄	3.1142		
Zn	0.7846	= ZnSO ₄	= 1.4368
	<hr/> 5.0398		<hr/> 5.0398

The Fe₂O₃ is a yellowish red precipitate in the flask. After filtering, it was added to the residue. It is believed to be a hydrated iron oxide such as limonite, since the amount of water found in the residue is only 10 per cent. less than what is theoretically required for the amount of Fe₂O₃ (0.7343) to be limonite. The results given above, show: (1) the partial oxidation of sphalerite due to free access of air; (2) that the ferrous sulphate may, under the influence of air, be oxidized to hydrated ferric oxide, and enable more sphalerite to be oxidized by giving up its sulphoxygen ions to the latter, as evidenced by the free sulphur recalculated in the mineral component column for the residue; (3) that ferrous sulphate does not oxidize sphalerite—the analysis of (a)—but if it has free access to air, it may form hydrated ferric oxide and subsequently oxidize all the Zn into sulphate, leaving its sulphur in the free state, if the mixture has been allowed to stand long enough in the air; (4) that the oxidation of ferrous to ferric by air in contact with sphalerite is probably exhibited, as in the case of pyrites with sphalerite;¹⁹ and (5) that, with

* This figure is calculated from the formula of limonite, assuming that the Fe is in the form of that mineral.

¹⁹ V. H. Gottschalk and H. A. Buehler. *Economic Geology*, vol. vii, pp. 22, 28 (1912).

sufficient time, all ZnS will go into solution as sulphate, and all the ferrous iron will oxidize to ferric oxide, which in the presence of water yields limonite or some other iron hydrate.

2. *The Solubility of Sphalerite in Salt Solutions of Silver, of Copper and of Lead.*—That zinc can be replaced or carried from its mineral in form of sulphide into solution to form a salt by the replacing action of a number of metallic elements in the form of their respective salts in solution has long been known to physical chemists. In 1888, Schuermann²⁰ established a series in which he showed that on application of heat the zinc as sulphide is dissolved in a solution of a soluble salt of any metal higher in the series, and that zinc can be precipitated as sulphide from a solution of its salt at the expense of a sulphide of any metal lower in the series. The solutions used in his experiments were all sulphates, nitrates, or chlorides. The following series was established in the order of the desire to combine with sulphur:

1, Pt; 2, Hg; 3, Ag; 4, Cu; 5, Bi; 6, Cd; 7, Sb; 8, Sn; 9, Pb; 10, Zn; 11, Ni; 12, Co; 13, Fe; 14, As; 15, Tl; 16, Mn. The first nine metals may be considered as dissolving agents and the last six as precipitants of the zinc sulphide. The reaction may be expressed in one case, thus: $\text{MSO}_4 + \text{ZnS} \rightarrow \text{ZnSO}_4 + \text{MS}$; and in the other case thus: $\text{MS} + \text{ZnSO}_4 \rightarrow \text{ZnS} + \text{MSO}_4$; M representing any metal which either precedes or follows zinc in the series.

The metals whose minerals are commonly found in geological association with those of zinc, are iron, copper, cadmium, lead, and occasionally silver. All these metals except the first precede zinc in Schuermann's series. Electrolytically, however, all of them can be displaced by zinc, since the electromotive force of the current required to deposit each metal is less than that for the metal preceding in the electromotive series.²¹

Some results of scientific interest might be obtained by treating the sulphide of zinc electrolytically in a salt solution of any metal preceding zinc in Schuermann's series; but this is outside our present purpose. It is not without interest, however, to consider the other possible reasons for the solubility of zinc sulphide in the solution of a salt of any one metal above zinc in this series, without attributing the reaction entirely to its affinity for sulphur, as did Schuermann. With this in mind, series of experiments were undertaken at different temperatures, degrees of concentration and time limits, in order to ascertain their relative importance, each to the other, and to observe in each case what change would take place under ordinary pressures. For this purpose two sulphates and one nitrate were taken—the three metals of which, silver, copper and lead, having, according to Schuermann, great affinity for sulphur.

²⁰ Ernest Schuermann. Über die Verwandtschaft der Schwermetalle zum Schwefel. *Liebig's Annalen der Chemie*, vol. cxxvii, p. 326 (1888).

²¹ Alex. Smith. *General Inorganic Chemistry*, p. 362.

Throughout the series of experiments a mixture of definite quantity in equal or less molecular ratio of each solution was used. In each case the mixture was subjected to various temperatures for a given time, and to a definite temperature for different periods of time, until the desired reaction was obtained. The solution of each metallic salt was kept at a uniform temperature before the beginning of each experiment, and the sphalerite used in all these experiments, unless indicated otherwise, was ground until it passed a 50-mesh sieve.

Experiment 4.—This comprised three tests, as follows:

(a) ZINC SULPHIDE WITH A SOLUTION OF SILVER SULPHATE

	Molecular Concentration				Temp., degree C.	Time	Reaction
	ZnS		Ag ₂ SO ₄				
	Grams	E. N.	C. C.	Norm.			
1	0.4873	$\frac{N}{10}$	100	$\frac{N}{25}$	15	4 hr.	None
2	0.2436	$\frac{N}{20}$	100	$\frac{N}{25}$	25	4 hr.	Slight
3	0.1950	$\frac{N}{25}$	100	$\frac{N}{30}$	30	4 hr.	Slight
4	0.1620	$\frac{N}{30}$	100	$\frac{N}{35}$	45	5 min.	Complete
5	0.1220	$\frac{N}{40}$	100	$\frac{N}{40}$	45	5 min.	Complete
6	0.1220	$\frac{N}{40}$	100	$\frac{N}{50}$	50	4 min.	Complete
7	0.4873	$\frac{N}{10}$	100	$\frac{N}{25}$	40	6 min.	Complete
8	0.4873	$\frac{N}{10}$	100	$\frac{N}{25}$	35	25 min.	Apparent, but not complete
9	0.4873	$\frac{N}{10}$	100	$\frac{N}{25}$	30	1 hr.	Slight
10	0.4873	$\frac{N}{10}$	100	$\frac{N}{25}$	25	3 hr.	Slight
11	0.4873	$\frac{N}{10}$	100	$\frac{N}{25}$	20	4 hr.	Slight
12	0.4873	$\frac{N}{10}$	100	$\frac{N}{25}$	0	20 hr.	None
13*	4.873	$\frac{N}{10}$	1,000	$\frac{N}{25}$	20	20 hr.	None
14*	4.873	$\frac{N}{10}$	1,000	$\frac{N}{25}$	30	12 hr.	Slight

E. N. = Equivalent normality. Norm. = Normality. Natural zinc sulphide and chemically pure silver sulphate were used.

* The sphalerite in this test was in its lump form without grinding.

(b) ZINC SULPHIDE WITH A SOLUTION OF COPPER SULPHATE
($\text{CuSO}_4 \cdot 5\text{H}_2\text{O}$)

	Molecular Concentration				Temp., degree C.	Time	Reaction
	ZnS		CuSO ₄				
	Grams	N. E.	C. C.	Norm.			
1	0.4873	$\frac{N}{10}$	100	$\frac{N}{10}$	15	4 hr.	None
2	0.2436	$\frac{N}{20}$	100	$\frac{N}{20}$	25	4 hr.	None
3	0.1950	$\frac{N}{25}$	100	$\frac{N}{25}$	30	4 hr.	Slight
4	0.1620	$\frac{N}{30}$	100	$\frac{N}{30}$	45	7 min.	Almost complete
5	0.1220	$\frac{N}{40}$	100	$\frac{N}{40}$	45	7 min.	Almost complete
6	0.1220	$\frac{N}{40}$	100	$\frac{N}{40}$	50	6 min.	Complete
7	0.4873	$\frac{N}{10}$	100	$\frac{N}{10}$	40	9 min.	Complete
8	0.4873	$\frac{N}{10}$	100	$\frac{N}{10}$	35	25 min.	Slight
9	0.4873	$\frac{N}{10}$	100	$\frac{N}{10}$	30	2 hr.	Slight
10	0.4873	$\frac{N}{10}$	100	$\frac{N}{10}$	25	5 hr.	Slight
11	0.4873	$\frac{N}{10}$	100	$\frac{N}{10}$	20	13 hr.	Slight
12	0.4873	$\frac{N}{10}$	100	$\frac{N}{10}$	0	13 hr.	None
13*	4.873	$\frac{N}{10}$	1,000	$\frac{N}{10}$	20	13 hr.	None
14*	4.873	$\frac{N}{10}$	1,000	$\frac{N}{10}$	30	15 hr.	Slight

E. N. = Equivalent normality. Norm. = Normality. Natural zinc sulphide and chemically pure copper sulphate were used.

* The sphalerite in this test was in its lump form without grinding.

(c) ZINC SULPHIDE WITH A SOLUTION OF LEAD NITRATE**

	Molecular Concentration				Temp., °C.	Time	Reaction
	ZnS		Pb(NO ₃) ₂				
	Grams	E. N.	C. C.	Norm.			
1	0.4873	$\frac{N}{10}$	100	$\frac{N}{10}$	15	4 hr.	None
2	0.2436	$\frac{N}{20}$	100	$\frac{N}{20}$	25	4 hr.	None
3	0.1950	$\frac{N}{25}$	100	$\frac{N}{25}$	30	4 hr.	Very slight
4	0.1620	$\frac{N}{30}$	100	$\frac{N}{30}$	45	13 min.	Complete
5	0.1220	$\frac{N}{40}$	100	$\frac{N}{40}$	45	13 min.	Complete
6	0.1220	$\frac{N}{40}$	100	$\frac{N}{40}$	50	11 min.	Complete
7	0.4873	$\frac{N}{10}$	100	$\frac{N}{10}$	40	18 min.	Complete
8	0.4873	$\frac{N}{10}$	100	$\frac{N}{10}$	35	30 min.	Slight
9	0.4873	$\frac{N}{10}$	100	$\frac{N}{10}$	30	3 hr.	Slight
10	0.4873	$\frac{N}{10}$	100	$\frac{N}{10}$	25	7 hr.	Slight
11	0.4873	$\frac{N}{10}$	100	$\frac{N}{10}$	20	14 hr.	Slight
12	0.4873	$\frac{N}{10}$	100	$\frac{N}{10}$	0	14 hr.	None
13*	4.8730	$\frac{N}{10}$	1,000	$\frac{N}{10}$	20	14 hr.	None
14*	4.8730	$\frac{N}{10}$	1,000	$\frac{N}{10}$	30	20 hr.	Very slight

E. N. = Equivalent normality. Norm. = Normality. Natural zinc sulphide and chemically pure lead nitrate were used.

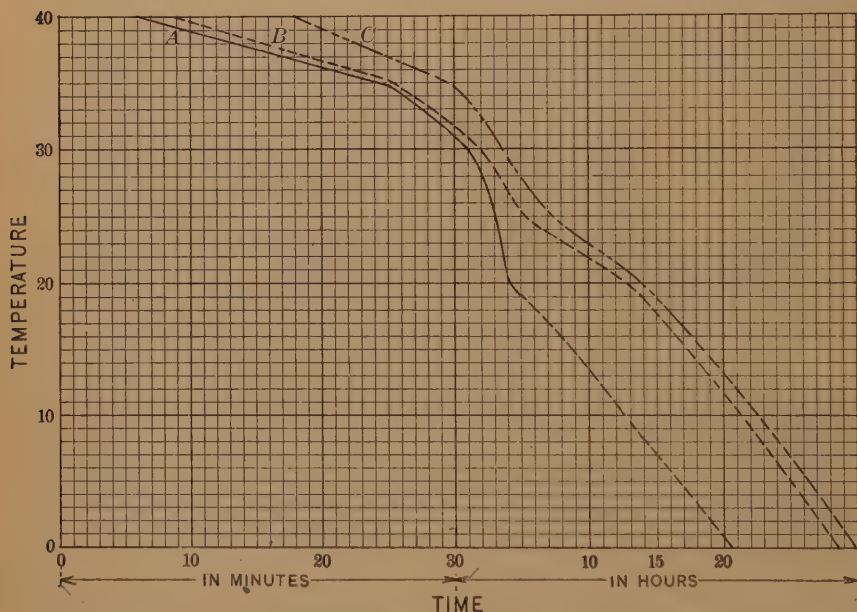
In explanation of the results in the foregoing three tables, it should be mentioned that the precipitated sulphides of Ag, Cu, and Pb were all amorphous and that all were qualitatively determined. No attempt was made for quantitative determination.

From these results it can be readily seen that temperature alone has played a most important rôle in the solubility of ZnS at its various mol-concentrations. This is best brought out by numbers 6 to 14 inclusive

* The sphalerite in this test was in its lump form without grinding.

** It was also tried with PbSO₄, but this compound is exceedingly difficult to dissolve, so Pb(NO₃)₂ was employed instead.

in each case. It is also interesting to note from a scientific point of view, that the temperature of 45°C. seems to be a critical one in all cases. The differences in the amount of ZnS and in the concentration of the reagent do not exert any appreciable influence upon the time required to complete the reaction. The differences in time required for the solution of ZnS in each of these different solutions at different concentrations is apparently due to the difference of their metallic sulphides either in potential values or in solubility products, or both. The single potentials



From 0 to 30 every small square represents one min.; from 30 to 20 to the right every small square represents one hr.

Curve A represents the reaction between ZnS and Ag_2SO_4 .

Curve B represents the reaction between ZnS and CuSO_4 .

Curve C represents the reaction between ZnS and $\text{Pb}(\text{NO}_3)_2$.

The broken part in each curve represents the unknown time required at a temperature between 20° and 0°C. Their concentrations are indicated in their respective tables.

FIG. 1.

of the metals, Ag, Cu and Pb, in solution saturated with their sulphides, as determined by Zengelis²² are as follows:

Ag in Ag_2S — 0.34 volts

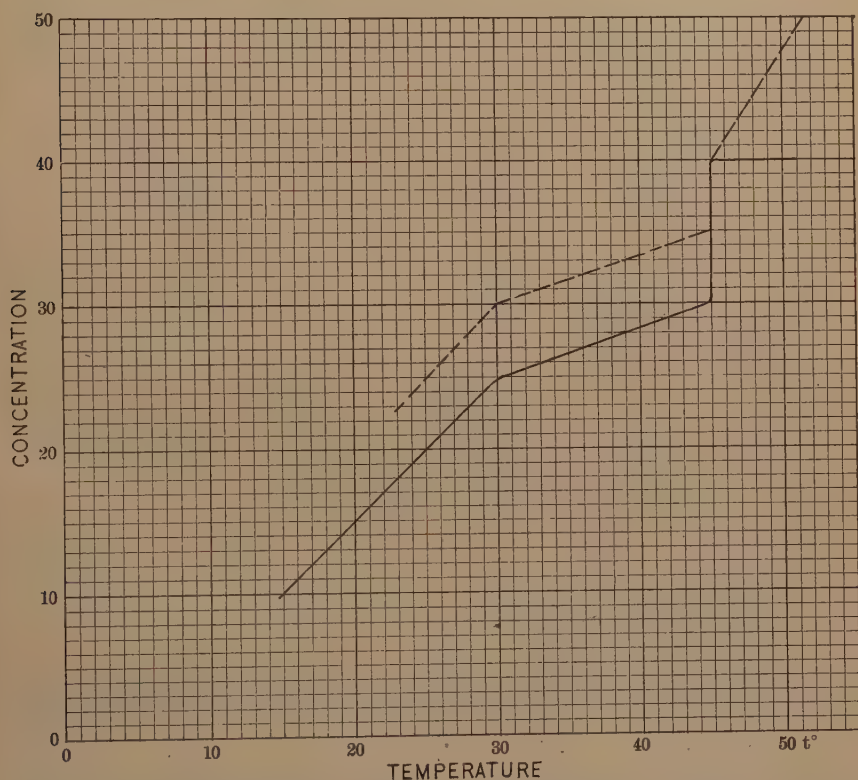
Cu in Cu_2S — 0.203 volts

Pb in PbS — 0.20 to 0.06 volts

The writer is unable to find any information concerning the exact potential value of zinc under similar conditions; but it is readily inferred

²² K. Zengelis. *Ueber die electromotische Kräfte unlöslicher und komplexer Salze.*

that the potential of zinc in zinc sulphides would be lower in comparison with that of any of these three metals from the fact that their sulphides can be precipitated from their salt solutions in contact with zinc sulphide at a definite temperature, as shown by the above experimental results. Their solubility product doubtless plays an equally important part in these reactions. The fact that less soluble compounds must be thrown down in presence of more soluble ones is too well known to be doubted



The dotted line represents the behavior of Ag_2SO_4 solution with ZnS . The continuous black line represents the general behavior of CuSO_4 and $\text{Pb}(\text{NO}_3)_2$ solutions with ZnS .

Note the influence of these three solutions on ZnS at the temperature of 45° irrespective of their concentrations.

FIG. 2.

Fig. 1 is intended to show the time required for the reaction at various temperatures between each of these three solutions and ZnS .

Fig. 2 illustrates the general behavior of ZnS with two different salt solutions of these three metals at different concentrations and temperatures, irrespective of the time required for the reaction in each case.

The above results may help to account for the fact that when zinc occurs in paying quantities as sulphide with the sulphide of Ag, Cu, or

Pb, the zinc always appears most abundantly only beneath the horizon of the minerals of the latter three, if they occur contemporaneously in the same horizon. It must not be understood that sphalerite does not occur side by side with galena or chalcopyrite or argentiferous galena. When they do occur together, it simply means there was enough sulphur to take care of all of them. This relation is particularly well shown in the occurrence of zinc-lead deposits in the Mississippi Valley, where, if galena is the chief ore, ZnS occurs only beneath the richest horizon of galena; if ZnS is the principal ore, PbS is found only in the upper portion of the orebody, as is the case with the ores in Joplin, Mo. The zinc mines at Butte²³ at the present time also show this to be true. In former times, silver sulphide occurred and was mined in the upper horizon. At present, blende is mined beneath that horizon. It is not unlikely that in these mines at Butte, copper sulphide may also occur at a greater depth in the future, but this would simply mean that the occurrences represent different periods of deposition.

With these geological observations on the one hand, and chemical relations on the other, there can be no doubt that ZnS , if present in solution together with sulphides of Pb, Cu, or Ag, is the last mineral to be formed in nature.

B. Formation of Zinc Carbonates

3. *The Reactions of Carbonates and Bicarbonates With Solutions of Zinc Salts.*—From the fact that the molar solubility of ZnCO_3 is much lower than that of ZnSO_4 , or ZnCl_2 , and that ZnS is often replaced in nature by ZnCO_3 , it is readily inferred that the substances in an orebody, capable of changing ZnS to ZnCO_3 , or precipitating ZnCO_3 from a solution in which Zn is held as a sulphate or chloride, are largely the soluble normal carbonates and bicarbonates.

The normal carbonates of sodium and potassium and bicarbonates of sodium and calcium were used for the following experiments to demonstrate this point.

Experiment 5.—On the precipitation of ZnCO_3 from its soluble compounds.

(a) Normal carbonate of sodium: 400 c.c. $\frac{\text{N}}{50}$ solution of ZnSO_4 was mixed with an excess of Na_2CO_3 solution of same normality. A flocculent precipitate formed immediately. Similarly 400 c.c. $\frac{\text{N}}{50}$ solution of ZnCl_2 was mixed with an excess of Na_2CO_3 solution and a precipitate likewise formed immediately. Each of these mixtures ($\text{ZnSO}_4 + \text{Na}_2\text{CO}_3$ and $\text{ZnCl}_2 + \text{Na}_2\text{CO}_3$) was then warmed and each precipitate was filtered and washed until the washings did not show the presence of Cl or SO_4 .

²³ Learned orally from Prof. J. F. Kemp, who has just visited some of the zinc mines at Butte, Mont.

The precipitate from $(\text{ZnSO}_4 + \text{Na}_2\text{CO}_3)$ weighed 0.4423 g. and that from $(\text{ZnCl}_2 + \text{Na}_2\text{CO}_3)$ 0.4322 g. Both were amorphous and each yielded the following after being dried in the oven and cooled in the desiccator:

The precipitate from the reaction $\text{ZnSO}_4 + \text{Na}_2\text{CO}_3$:

Chemical Components	Grams	Per Cent.	Molecular Weight	Molecular Ratio
ZnO.....	0.33320	75.34	81.40	0.9255
CO ₂	0.06003	13.57	44.00	0.3084
H ₂ O.....	0.04912	11.21	18.00	0.6227
	0.44235	100.12		

Molecular proportions:

$$0.9255 : 0.3084 : 0.6227 = 3 : 1 : 2 = 3\text{ZnO} \cdot \text{CO}_2 \cdot 2\text{H}_2\text{O}.$$

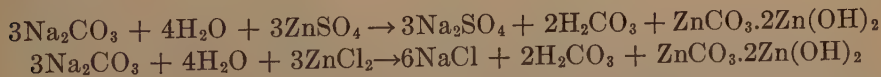
The precipitate from the reaction $\text{ZnCl}_2 + \text{Na}_2\text{CO}_3$:

Chemical Components	Grams	Per Cent.	Molecular Weight	Molecular Ratio
ZnO.....	0.32560	75.33	81.40	0.9252
CO ₂	0.05866	13.59	44.00	0.3088
H ₂ O.....	0.04799	11.10	18.00	0.6166
	0.43225	100.02		

Molecular proportions:

$$0.9252 : 0.3088 : 0.6166 = 3 : 1 : 2 = 3\text{ZnO} \cdot \text{CO}_2 \cdot 2\text{H}_2\text{O}.$$

The above results clearly show that each of these precipitates represents, not a normal zinc carbonate, but a basic zinc carbonate such as hydrozincite. Its reaction may be represented by some such equation as this:



Moreover, Dana, in his *System of Mineralogy*, suggested this formula for hydrozincite. According to him, the constituents of a hydrous zinc carbonate from Auronz, were as follows: ZnO, 73.21; CO₂, 14.55; and H₂O, 11.38 per cent.

In another experiment, the reactions between Na_2CO_3 and ZnSO_4 solutions gave likewise a precipitate of basic zinc carbonate, but it was found to contain only one molecule each of ZnCO_3 and $\text{Zn}(\text{OH})_2$. This difference probably indicates that the formation of basic zinc carbonate varies in composition according to conditions.²⁴

²⁴ Alex. Smith. *General Inorganic Chemistry*, p. 649.

The precipitate obtained from the reaction of potassium carbonate with ZnSO_4 or ZnCl_2 was identical in appearance, but no analysis was made.

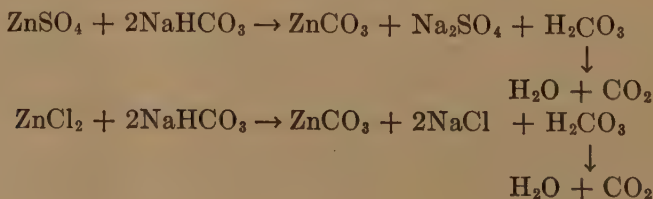
Duplicates of this experiment between ZnSO_4 and Na_2CO_3 were performed at the same time and under same conditions; but instead of filtering off the precipitate, carbon dioxide was allowed to pass directly into the mixture for half an hour to one hour. The mixture (formerly precipitate and liquid) became in this time a clear solution showing that ZnCO_3 is not stable in presence of free H_2CO_3 . This agrees with the fact that in the waters of zinc mines, where zinc carbonates occur, no H_2CO_3 has been reported.

(b) 1. Bicarbonate of Sodium (NaHCO_3): To an excess of this bicarbonate solution was added 400 c.c. of $\frac{\text{N}}{50}$ ZnSO_4 solution. The mixture soon became milky and a very fine precipitate was formed. Some CO_2 gas appeared during the precipitation. In the same way ZnCl_2 was treated with NaHCO_3 and the same kind of precipitate was formed. Each mixture was heated almost to boiling in order to drive off the excess of CO_2 . Each precipitate was separated from its liquid by filtration, and was washed until no sign of SO_4 or Cl was shown. The precipitate in each case was amorphous.

The weight of the precipitate from ZnSO_4 was 0.5112 g., and that from ZnCl_2 , 0.4873 g. After being dried in the oven and cooled in the desiccator they yielded the results given below:

The precipitate from the reaction between ZnSO_4 and NaHCO_3			The precipitate from the reaction between ZnCl_2 and NaHCO_3		
	Grams,	Per Cent.		Grams,	Per Cent.
ZnO	0.3302 =	64.6		0.3164 =	64.94
CO_2	0.1798 =	35.18		0.1712 =	35.14
H_2O	trace			none	
Total.....	0.5100	99.78	Total.....	0.4876	100.08

The substance is evidently anhydrous carbonate of zinc and this reaction may be represented by the following simple equation:



Theoretically anhydrous carbonate of zinc contains: ZnO , 64.8; CO_2 , 35.2 per cent.

(b) 2. Bicarbonate of Calcium: 1.001 g. of finely powdered pure limestone (an equivalent of $\frac{N}{50}$) were placed in 1,000 c.c. water, into which carbon dioxide was subsequently allowed to pass until the powdered mineral entirely disappeared, being converted into the more soluble bicarbonate. The reaction may be represented by the following equation:



400 c.c. of this lime solution were mixed with a slight excess of $\frac{N}{50}$ solution of ZnSO_4 in one flask and ZnCl_2 in the other. Each mixture immediately became milky and a very fine precipitate formed. The liquid was almost neutral. After being heated to boiling each precipitate was filtered off and washed until the last few drops of washing water no longer showed the presence of SO_4 or CO_3 ions. Then each was dried in the steam oven. They were weighed, after they had been allowed to cool in the desiccator, and yielded respectively:

A. The precipitate from $\text{Ca}(\text{HCO}_3)_2 + \text{ZnSO}_4$ weighed 0.4581 g.

	Grams	Per Cent.	Molecular Weight	Molecular Ratio	Molecular Proportion
CaO	0.11290	24.63	56.1	0.4389	1
CO_2	0.17862	38.98	44.0	0.8700	2
ZnO	0.10650	36.34	81.4	0.4287	1

B. The precipitate from $\text{Ca}(\text{HCO}_3)_2 + \text{ZnCl}_2$ weighed 0.4272 g.

	Grams	Per Cent.	Molecular Weight	Molecular Ratio	Molecular Proportion
CaO	0.11070	25.90	56.1	0.4616	1
CO_2	0.10743	39.19	44.0	0.8906	2
ZnO	0.14910	34.90	81.4	0.4287	1

Their mineral components according to the amount of the above chemical analyses, may be represented respectively:

(A)	(B)
$\text{CaO} \cdot \text{CO}_2$ or $\text{CaCO}_3 = 0.2014$ g.	$\text{CaO} \cdot \text{CO}_2$ or $\text{CaCO}_3 = 0.1970$ g.
$\text{ZnO} \cdot \text{CO}_2$ or $\text{ZnCO}_3 = 0.2566$ g.	$\text{ZnO} \cdot \text{CO}_2$ or $\text{ZnCO}_3 = 0.2320$ g.

From this it appears that the compound consists of carbonates of calcium and zinc. From the amount of calcium carbonate used at the

start, before its conversion of $\text{Ca}(\text{HCO}_3)_2$, and from the amount of the mineral component of the same compound, it is readily seen that the amount of Ca present in 400 c.c. of $\frac{N}{50}$ $\text{Ca}(\text{HCO}_3)_2$ solution is 0.1614 g. and that the metal recovered in its original form of CaCO_3 by interaction with ZnSO_4 is 0.0808 g. or a little over half of its original amount.

It is also theoretically known that one molecule of $\text{Ca}(\text{HCO}_3)_2$ in solution is able to precipitate an equivalent of ZnCO_3 , and to produce a complete reformation of CaCO_3 , provided they are not affected by the byproduct H_2SO_4 , as indicated in the equation:



But this hardly appears to be correct, as shown by the amount of the mineral components which is only about half of what this equation demands. Suffice it to say that the formation of the "secondary" H_2SO_4 (or HCl , if ZnCl_2 is used instead of ZnSO_4) must be used up in its reaction with the chemically precipitated ZnCO_3 , and CaCO_3 , thus converting part of both of these precipitates into ZnSO_4 and CaSO_4 on the one hand, and giving rise to the formation of H_2CO_3 on the other until the new equilibrium is reached. These relations may be best shown by the following equation:



As soon as this condition is reached the rest of the CaCO_3 and ZnCO_3 will be able to remain as precipitates, since these carbonates are quite stable in the solution of their sulphates. It is also possible that the carbonates of Zn and Ca may co-exist as one compound, analogous to dolomite, whose formula is ordinarily written: $\text{MgCO}_3 \cdot \text{CaCO}_3$.

The H_2CO_3 shown in equation (2) doubtless breaks up into H_2O and CO_2 . These relations are further proved by the qualitative test of the filtrate which, after being heated to boiling for 15 min., showed only the presence of Ca, Zn and SO_4 ions.

These reactions may help to account for the occurrence of zinc carbonate in, and in contact with, limestone in nature, as in the Bertha Mines of Virginia.

Dr. Watson²⁵ has suggested that the calcium carbonate must be first converted into soluble calcium bicarbonate due to "the interaction of the percolating atmospheric waters containing carbon dioxide in solution on limestone," before the reaction to form zinc carbonate takes place. This is in full accord with the writer's experience in the laboratory. He further suggested that in the formation of normal zinc carbonate, under natural conditions, the compound is probably in the form of basic carbon-

²⁵ T. L. Watson. *Bulletin No. 1, Geological Survey of Virginia*, p. 44 (1905).

ate as an "intermediate step" towards the formation of normal zinc carbonate from the reactions between zinc sulphate and calcium bicarbonate. But the writer's laboratory experience tends to indicate that normal zinc carbonate only is formed if the reagent is a bicarbonate, whereas a basic carbonate of zinc only is precipitated in varying composition if the reagent is a normal carbonate.

The precipitates from duplicates were not filtered, but directly treated with carbon dioxide, which was allowed to pass into the mixture for about three-quarters of an hour. At the end of this period the precipitates had entirely disappeared and the liquid had become clear, indicating that in presence of free H_2CO_3 the normal carbonate of zinc cannot exist, probably just as the calcium carbonate became soluble, forming bicarbonate, under the same condition, as shown in the preparation of it for the above experiments.

Similar experiments were performed with chemically made sulphide of zinc and with natural sulphide from Joplin, Mo. In these experiments 0.4873 g. (the equivalent of $\frac{N}{100}$) of each of these sulphides was treated in 1,000 c.c. water and mixed with a double amount of $\frac{N}{100}$ solution of normal sodium carbonate in one flask, and of bicarbonates of sodium and calcium in the other, as before. No evidence of appreciable change in each case was noted in the course of two months. Carbon dioxide was then passed into each of these mixtures for an hour, but evidence of change was equally absent in each case. This suggests that direct oxidation of sphalerite to zinc carbonate is far from being realized under laboratory conditions.

Experiment 6a.—On the Conversion of Zinc Carbonate from Zinc Sulphide. These experiments were repeated, but only the natural sulphide of zinc was used, with modifications in the conduct of the experiments. 100 g. of coarsely crushed (not powdered) sphalerite was spread in layers in a large glass jar (A, Fig. 3), 9 in. high and 4 in. in diameter, with alternate layers of various sizes of sandstones, shales and limestones above and below the sulphide layers. The entire charge filled the jar three-quarters full. A solution of $\frac{N}{50}$ CuSO_4 was first allowed to flow slowly into this jar through a glass tube (T_1) and as soon as the solution filled the jar up to three-quarters of the depth, the solution was allowed to flow through another glass tube (T_2), which was fixed to reach downward one-fourth of the depth of the jar, into another bottle which served as a receiver (C) containing the excess of the solution of CuSO_4 , which was returned to the supplying bottle (B) from time to time. Altogether 18,000 c.c. of $\frac{N}{50}$ CuSO_4 solutions were used during the entire period, including fresh portions of this solution, added from time to time

to compensate for evaporation. This treatment was continued for a period of four months, under ordinary conditions of temperature and pressure.

At the end of this period the colorless solution and the blackish precipitate, formed during the operation in the jar (*A*), and the solutions in bottles (*B*) and (*C*), amounting by this time to about 3,400 c.c., were all emptied into a large flask. The sandstone, shale and limestone together

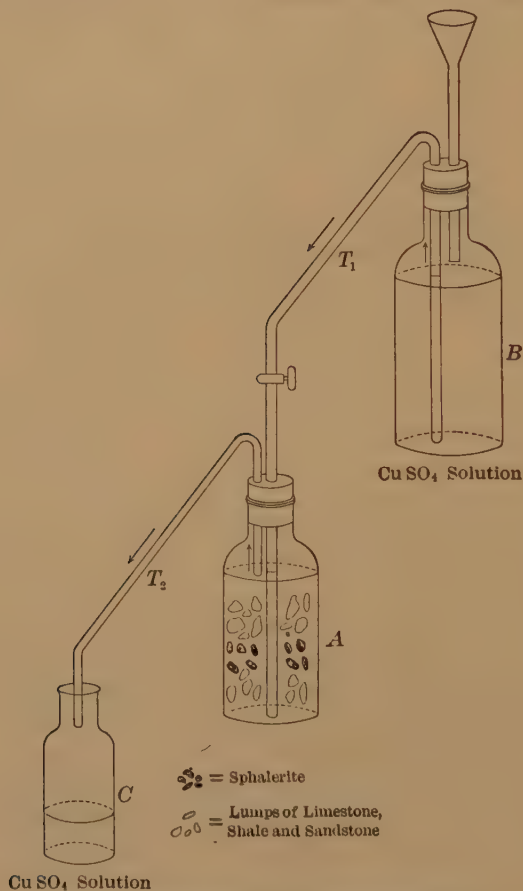


FIG. 3.—APPARATUS FOR EXPERIMENT 6a.

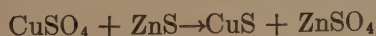
with the remaining lumps of sphalerite in jar (*A*), were carefully washed with water until the washing water was free from SO₄ ions. All the washings were added to the mixture.

The blackish precipitate was then filtered off from this mixture and washed until the washings showed no presence of SO₄ ions. These washings were also added to the filtrate which was then made up to 5,000 c.c. with distilled water.

It was the purpose of this experiment to ascertain in a quantitative way: (1) whether copper as sulphate in dilute solutions would attack the natural ZnS in contact with limestone, shale and sandstone; and (2) whether such reagent would also dissolve part of those rocks in the jar, apart from sphalerite, in the course of circulation. To determine these two points, definite portions of the filtrate and the precipitate, whose weight after being dried in the desiccator was found to be 17.2 g., and which contained some free sulphur, were taken for analysis. The filtrate, according to litmus paper tests, was slightly acidic or almost neutral; 500 c.c. of it were used in the analysis recorded below: The precipitate yielded:

				Amount in Precipitate and in Filtrate	Amount of Cu and SO ₄ Originally Present in Solution
	Grams	Grams	Grams	Grams	Grams
ZnO.....	2.1790	Zn	1.7520	1.7520
Cu.....	11.4400	Cu	11.4400	11.4400	11.4400
CO ₂	1.1790
S including free S..	5.7600	gangue	7.4662	7.4662
SiO ₂	0.1002
	20.6582		20.6582		
and from the fil- trate were ob- tained					
SiO ₂	0.0018	SiO ₂	0.0018
Al ₂ O ₃	0.00366	Al	0.00194
FeO.....
Fe ₂ O ₃	0.00844	Fe	0.00590
CaO.....	0.03673	Ca	0.02625
MgO.....	0.06253	Mg	0.03775
Na ₂ O.....	0.00824	Na	0.00612
K ₂ O.....	0.01250	K	0.01038
SO ₄	1.72810	SO ₄	1.72820	17.2820	17.2820
ZnO.....	1.24700	Zn	1.00200	10.0200
				47.9602	28.722

From the molecular weight, it is known that 18,000 c.c. $\frac{N}{50}$ CuSO₄ solution furnish about 11.448 g. Cu, and from the chemical reactions,



the amount of copper (11.440 g.) actually found in the precipitate would require, by calculation, about 5.76 g. sulphur to form CuS.²⁶ The amount

²⁶ The CuS is believed to have been reduced to Cu₂S and the free S in the precipitate is due to this reduction.

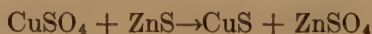
of zinc (1.7520 g.) was undoubtedly in the form of carbonate, precipitated from zinc sulphate, as may be inferred also from the presence and amount of carbon dioxide in the analysis. Thus by calculation, according to the above analysis, the precipitate contains the following:

	Grams		Grams
CuS	= 17.20	= Cu	11.4400
ZnCO ₃	= 3.358	= Zn	1.7520
SiO ₂	= 0.1002	= gangue	7.4662
	<hr/>		<hr/>
	20.6582		20.6582

The analysis of the filtrate, through the absence of Cu, points to its precipitation from solution, and the presence of an almost equivalent amount, molecularly speaking, of zinc in the solution as sulphate, indicates the oxidation of zinc from its sulphide. Lastly, the presence of other substances, which were absent in the mixture at the beginning of the experiment, indicates that a solution containing sulphate of copper (or a soluble salt of any other metal beneath hydrogen in the electromotive-force series) tends to carry many common rock-making minerals or components into solution and, as a consequence, neutralize or decrease its acidic character to a very appreciable extent.

It is also known from the molecular weight that 18,000 c.c. $\frac{N}{50}$ solution of CuSO₄ furnishes 11.448 g. Cu and 17.2818 g. SO₄.²⁷ Theoretically, a $\frac{N}{50}$ solution of cupric sulphate in one liter requires 0.654 g. Zn to displace all the Cu.

But from the fact that Cu is entirely absent in the filtrate, as shown by the above analysis, the theoretical amount of Zn oxidized into sulphate or required to replace the amount of Cu actually found in the precipitate, cannot be less than 11.76 g., as based on the reaction,



By deducting 11.76 g. (to satisfy the replacement of the Cu) from the amount of Zn found in the filtrate (10.02 g.) and in the precipitate (1.752 g.) there is an excess of 0.012 g. (11.772 - 11.76 = 0.012), which is about 0.1019 per cent. more than what is theoretically necessary for the amount of copper as found in the analysis of the precipitate. The difference in these values (between theoretical and practical calculations) is undoubtedly due to errors in manipulation.

The amount of SO₄ in the original solution as represented by CuSO₄ at start was 17.2818 g. (theoretically).²⁷ The amount recovered was

²⁷ In calculating the SO₄ or Cu from the sulphate of copper no water of crystallization was considered.

17.282 g. After deducting from the total amount of SO_4 (17.282 g.) the theoretical amount required for all the Zn as sulphate found in the filtrate, there remains 2.552 g. ($17.282 - 14.73 = 2.552$) which may be safely assigned to the other bi- and tri-metals found in the filtrate.

That the sulphide of zinc is oxidized by the solution of copper sulphate is practically certain. The remaining solution and the washings, chiefly containing Zn as sulphate, are now ready for the treatment as used in experiment 6b.

Experiment 6b.—The precipitation of ZnCO_3 from the oxidized solution of ZnS : 500 c.c. of this mixed solution was treated with an excess of $\frac{\text{N}}{50}$ solution of NaHCO_3 . A very fine precipitate was formed immediately. The whole mixture was gently heated almost to boiling; some CO_2 was noted passing off during the heating. The mixture was allowed to cool and the precipitate was separated from the liquid by filtration. The former was washed until the washings showed no sign of the presence of SO_4 . It was then dried in the steam oven and cooled in the desiccator before the precipitate was weighed (1.9884 g.) and analyzed. The following are the analytical results:

	Grams	Per Cent.
SiO_2	0.0019	0.095
CaO	0.0367 (= Ca, 0.02625)	1.842
MgO	trace
H_2O	none
CO_2	0.7105	35.260
ZnO	1.2470 (= Zn, 1.002)	62.560
	<hr/> 1.98838	<hr/> 99.757

From the above analysis, it may be safely assumed that all the chemical components present (except SiO_2) were in the form of carbonates, since by recasting for their carbonates it is found, as shown below, that the amount of CO_2 required by these by-metals (Zn and Ca) is 0.7028 g. Namely,

0.02625 g. Ca will form 0.0655 g. CaCO_3 with 0.0288 g. CO_2

1.0020 g. Zn will form 1.9210 g. ZnCO_3 with 0.6740 g. CO_2

0.7028 g.

The remaining CO_2 (0.0077 g.) is probably in combination with MgO ; but the latter is too small in amount to be considered. The amount of these mineral components as carbonates is obtained by calculation from their chemical components. Their mineral component proportion may be represented as follows:

	Grams	Per Cent.
CaCO ₃	0.0655	3.294
MgCO ₃	trace
ZnCO ₃	1.9210	96.61
		<hr/> 99.904

Since there is enough CO₂ to take care of all the zinc found, and no water is present, and SiO₂ is too small in amount to be considered as having combined with Zn in any way, the Zn compound is, beyond doubt, a normal carbonate.

In another experiment a mixture of dilute solutions of bicarbonate was allowed to pass through a jar containing sphalerite in the manner used for the experiments under 6a. The operation was continued for two months, but met with no success.

In the light of the above results, it seems quite certain that ZnCO₃ is not formed direct from its sulphide. The sulphide is first oxidized into zinc sulphate, and the latter may migrate a short or long distance, as the case may be, before the carbonation takes place. In the waters which percolate through strata in which ZnS (sphalerite) occurs, the chemically active compounds must be complex, and must be provided with oxidizing material of one kind or another. These waters will doubtless oxidize the sphalerite at an incipient stage to sulphate. The oxidized part will, according to existing conditions, probably be transported to some other place until it meets an alkaline solution or solutions of alkaline earths, which in nature mostly contain soluble carbonates. The deposition of ZnCO₃ will then take place, if the metals like Pb or Cu, which will form more insoluble carbonates than Zn, are either absent or very small in amount. These conditions will account for the fact that both ZnCO₃ and ZnS have been found in different horizons as if they had no relation to each other, as in the Circle Mines at Oronogo, Mo.²⁸ and in some zinc mines in Virginia.²⁹

In case an orebody of sphalerite is attacked by a neutral solution of sulphates and carbonates, the sphalerite may be first oxidized into sulphate, especially when the metals above zinc in Schuermann's series³⁰ are present as sulphates in the solution, and then immediately carbonated, without suffering the removal of any appreciable quantity by migration. This may help to account for the fact that sphalerite is sometimes closely associated with smithsonite, and the latter always grades downward into the former.

4. The Reaction of Zinc Sulphate Solution with Limestones.—The

²⁸ The writer had an opportunity in the summer of 1913 to visit some of the zinc mines in southwest Missouri, particularly in the Joplin district. All the writer's own field data were obtained at that time.

²⁹ T. L. Watson. *Bulletin No. 1, Geological Survey of Virginia*, p. 45 (1905).

³⁰ *Liebig's Annalen der Chemie*, vol. ccxlvii, p. 326 (1888).

*Experiment 7: With Crystalline Limestone**

Analysis of the Limestone					Analysis of the Solution	
Before Treatment			After Treatment†		Before Treatment	After Treatment
	Grams	Per Cent.	Grams	Per Cent.	Grams	Grams
SiO ₂	0.0427	0.6521	CaO	0.6063 = 1.472 CaSO ₄
Al ₂ O ₃	0.0234	0.3574	SO ₄ 2.597	2.5967
Fe ₂ O ₃	0.0183	0.2796	ZnO	1.3200 = 2.619
					or	ZnSO ₄ or
CaO.....	3.543	54.1200 = 96.54	2.9380	43.1200 = 76.93	Zn 1.768	1.0470
		CaCO ₃		CaCO ₃		
MgO.....	0.0193	0.2947 = 0.6155	0.0156	0.2290 = 0.4769		
		MgCO ₃		MgCO ₃		
CO ₂	2.800	42.7408	2.7966	41.0400		
ZnO.....			0.8800	12.9100 = 19.9000		
			or	ZnCO ₃		
Zn.....			0.7071			
	6.4467	97.1655		97.3069		

Experiment 8: With Impure Limestone

Analysis of the Limestone					Analysis of the Solution	
Before Treatment			After Treatment		Before Treatment	After Treatment
	Grams	Per Cent.	Grams	Per Cent.	Grams	Grams
Insoluble and organic matter.....	3.432	CaO	1.16
SiO ₂	2.406	MgO	0.2813
Al ₂ O ₃	3.512	SO ₄ 2.597	2.597
Fe ₂ O ₃	0.9223	ZnO 1.768
CaO.....	13.5200	34.26 = 61.12	12.380	31.37 = 55.94		
		CaCO ₃		CaCO ₃		
MgO.....	1.9680	4.987 = 10.42	1.687	4.275 = 8.933		
		MgCO ₃		MgCO ₃		
H ₂ O +.....	0.3117		
H ₂ O -.....	0.1947		
CO ₂	12.751	12.735		
MnO.....	trace		
Na ₂ O.....	0.1270		
K ₂ O.....	0.325		
ZnO.....		2.201	7.158 = 11.03		
Zn.....		1.750	ZnCO ₃		
	34.4697	71.54				

* This limestone was obtained by Dr. C. P. Berkey from Rockland. Dr. Berkey gave CaCO₃, 96.45, and MgCO₃, 0.12 per cent. in one case; and CaCO₃, 94.9 and MgCO₃, 0.20 per cent. in the other.

† H₂O and organic matter were not determined. Only carbonates determined.

solution of zinc sulphate used in the following experiments is all $\frac{N}{20}$, that is, 4.365 g. $ZnSO_4$ or 1.768 g. metallic zinc in every 1,000 c.c. The limestones used were analyzed before and after the treatment, which in each case was kept going for a period of four months in a closed bottle, to which air was occasionally admitted through a side tube. In each case a definite weight of limestone was treated with 1,000 c.c. $\frac{N}{20}$ solution of $ZnSO_4$. At the end of four months the limestone in each case was found to have been covered with a white thin coat which, upon drying, appeared to be somewhat powdery, and which was $ZnCO_3$ as shown in the respective analyses recorded on preceding page.

From the above analyses, it is shown that in the case of crystalline limestone, only two-thirds of the zinc was precipitated as $ZnCO_3$, while with the impure limestone, zinc is entirely absent in the sulphate solution after the treatment. These apparent differences from the treatment of two different limestones with the same solution of $ZnSO_4$ are believed to be due to the presence of organic matter, which is common in impure limestone.

It is further shown that carbonates of both magnesium and calcium can be replaced by zinc under the conditions described above.

In another experiment with the impure limestone, minute particles of what appeared to be ZnS could be seen with the aid of a powerful lens. Such precipitation of the zinc as sulphide, by commingling of the oxidizing and reducing solutions in limestones, is not only possible, but also probable, according to some geologists, notably Dr. Buckley,³¹ in his studies of the origin of zinc sulphide in the Joplin district. The evidence that these minute particles were ZnS , was, however, not conclusive, for the reason that it was difficult to separate them from the whitish coating of zinc carbonate. An attempt was made to prepare thin sections, but the replaced material was too soft for grinding.

C. Peculiar Conditions Affecting the Formation of Zinc Silicate (Calamine) and Zinc Carbonate

5a. *The Effect of Carbon Dioxide upon the Basic Silicate and the Carbonates of Zinc in Water, and its Bearing upon the Conditions of their Precipitation.*—Hydrous silicate of zinc³² has long been known to be soluble in water in presence of CO_2 gas, but no one has attempted to explain in the light of modern chemistry the bearing of the latter upon the formation of this silicate and the association of the silicate with carbonate of the

³¹ E. R. Buckley. *Types of Ore Deposits*, pp. 103 to 132.

³² In 1849, Victor Monheim made his quantitative determination of zinc silicate by dissolving it in water in presence of carbon dioxide. *Verhandlungen des Naturhistorischen Vereins*, vol. vi, pp. 1 to 24.

same metal. The following experiments were undertaken in this connection.

Experiment 9a.—5 g. of pure calamine from the vicinity of Chihuahua, Mexico, secured from the collection of ores in the Department of Geology at Columbia University, was powdered until it passed 100-mesh sieve. This amount was mixed with 1 liter of water in a 2-liter flask. A stream of carbon dioxide was allowed to pass through for 80 hr., with occasional shaking of the contents. The mixture was filtered and a residue collected and washed before it was dried in the oven. The washings were added to the filtrate, which was perfectly clear. After being dried in the oven and cooled in the desiccator, the residue was weighed at 3.48 g. For convenience it is designated as first residue.

The filtrate was heated to boiling for about half an hour in order to drive off the CO_2 completely. A fine precipitate came down, which was also filtered, washed and dried in the oven. Its weight was found to be 1.462 g. It is designated as second residue. There is a loss of 0.012 per cent. due to manipulation.

By analysis these residues yielded respectively:

	First Residue		Second Residue	
	Grams	Per Cent.	Grams	Per Cent.
ZnO.....	2.3480	67.48	0.9868	67.50
SiO ₂	0.8714	25.03	0.3661	25.03
H ₂ O.....	0.2597	7.463	0.1091	7.461
		99.973		99.991

That the hydrated silicate of zinc is soluble in water in presence of enough carbon dioxide, and that the same silicate can be reconverted into its solid form from such solution by raising the temperature, there is no doubt. It is also believed that the carbon dioxide does not combine with zinc to form some such soluble compound as bicarbonate of zinc, for the H_2O in the zinc silicate is basic,³³ and if bicarbonate were formed, zinc carbonate would come down as precipitate when the CO_2 was driven off by heating. Thus the carbon dioxide merely exerts its influence as a catalytic agent in carrying the silicate of zinc into solution, without affecting any change in its molecular combination.

Experiment 9b.—The next question is whether or not an impure calamine, that is a mixture of zinc silicate and zinc carbonate with carbonates of some other bi-metals will also become soluble in water in excess of carbon dioxide as shown in each individual case. For this purpose a piece of impure calamine, obtained from a miner in Joplin, Mo., was chosen for the experiment.

In order to know the exact contents, 3.259 g. of the powdered material from the specimen were analyzed, with the following results:

³³ Dana. *System of Mineralogy*, Sixth Edition, p. 548.

Chemical Components			Mineral Components		
	Grams	Per Cent.		Grams	Per Cent.
SiO ₂	0.5622	17.300	Calamine	2.2470	69.130
Fe ₂ O ₃	0.0377	1.160	Zinc carbonate	0.7632	23.480
FeO.....	trace	Calcium carbonate	0.1378	4.240
CaO.....	0.0772	2.375	Magnesium carbonate.....	0.0786	2.418
MgO.....	0.0376	1.157	Limonite	0.0443	1.363
CO ₂	0.3673	11.310
ZnO.....	2.0110	61.870
H ₂ O.....	0.1758	5.409
Na&K.....	trace
MnO.....	trace
P ₂ O ₅	none
	3.2688	100.581		3.2709	100.631

An equal amount (3.259 g.) of this material was taken for the treatment in the experiment, which was in every respect the same as in the last case. At the end of the experiment, the mixture was filtered. Upon boiling the filtrate, as before, a heavy precipitate collected. After being dried in the oven and cooled in the desiccator, it was found to weigh 2.522 g. and yielded:

Chemical Components				Mineral Components			
	Grams	Per Cent.	Recalculated Per Cent.		Grams	Per Cent.	Recalculated Per Cent.
SiO ₂	0.4109	16.2900	12.6400	Calamine.....	1.6412	65.0700	50.4900
Fe ₂ O ₃	0.0107	0.4243	1.2650	Zinc carbonate.	0.7311	28.9900	22.5000
FeO.....							
CaO.....	0.0411	1.6290	1.0490	CaCO ₃	0.0734	2.9110	2.2500
MgO.....	0.0341	1.3520	0.3293	MgCO ₃	0.0713	2.8280	2.1940
CO ₂	0.3261	13.2000	10.0300	Limonite.....	0.0126	0.4997	1.3377
ZnO.....	1.5825	63.2800	48.7000
H ₂ O.....	0.1243	4.9280	3.8230
	2.5297	99.1033	77.8363		2.5296	100.2987	77.8297

The chemical components were calculated, by recasting, into mineral components for the purpose of finding the exact amount of each mineral component dissolved in water in presence of the carbon dioxide. Its percentage was then recalculated (column 3) on the basis of the original weight of the material used in the experiment.

By comparing the figures in column 3 with the mineral percentage in the analysis of the specimen it can be readily seen: (1) that in the presence of carbon dioxide, nearly all the carbonates in the specimen were carried

- into solution; (2) that the zinc carbonate was almost entirely dissolved; and (3) that a much greater percentage of zinc silicate was dissolved from the impure specimen than from the pure one used in experiment 7a. It is, therefore, plausible that the presence of the impurities in the specimen, particularly ZnCO_3 , must have exerted some influence upon the solution of the zinc silicate. It is at least suggestive that the molecules of ZnCO_3 , which were present with those of the hydrous silicate of zinc in the material used, would attract molecule by molecule the components of the silicate in water under the influence of carbon dioxide, but would not necessarily effect any change in their respective molecules or molar-volumes, since their equilibrium remains the same in the aqueous state under the above conditions.

5b. *The Reactions of Solutions of Soluble Silicate with Calamine and Smithsonite in Solution in Presence of Carbon Dioxide.*—Since the excess of carbon dioxide can hold both silicate and carbonate of zinc in solution, the question arises as to what peculiar effect will be produced when a silicate solution is introduced into such a mixture. That is to say, whether the basic silicate can be brought about by means of a soluble silicate solution from a solution of zinc carbonate and zinc silicate made by dissolving the latter two under the influence of carbon dioxide.

It has long been known that silicon dioxide may replace carbon dioxide when heated with an alkaline carbonate,³⁴ and inasmuch as this property of silicon dioxide may have direct bearing on the origin of zinc silicate in association with zinc carbonate such as frequently found in zinc mines, the study of its precipitating capacity was undertaken.

Experiment 10.—4 g. of finely powdered calamine obtained from Chihuahua, Mexico, and 4 g. of chemically pure zinc carbonate were mixed together. The mixture (8 g. altogether) was then digested in 1,000 c.c. water under the influence of carbon dioxide, which was allowed to pass slowly (one bubble for every three seconds) through the mixture for a period of 50 hr. At the end of this period, the mixture was filtered and a residue, which for clearness, will be designated as residue X, was removed. It was dried in the steam oven and weighed 3.9692 g. By analysis the residue X, yielded:

	Grams	Per Cent.	Molecular Weight	Molecular Ratio	Molecular Proportion
SiO_2	0.5380	13.550	60.4	0.2244	3
CO_2	0.6387	16.090	44.0	0.3656	
H_2O	0.1603	4.033	18.0	0.2241	
ZnO	2.6310	66.300	81.4	0.8145	3
	3.9680	99.973			

³⁴ Alex. Smith. *General Inorganic Chemistry*, p. 519.

The filtrate, after the removal of the residue X, was treated with an excess of dilute solution of Na_2SiO_3 . A gelatinous precipitate, designated as precipitate Y, immediately collected and was filtered off, dried in the steam oven, and weighed, after being cooled in the desiccator, 2.3763 g.

As shown by the analytical results recorded below, it is certain that the gelatinous precipitate is not calamine ($2\text{ZnO} \cdot \text{SiO}_2 \cdot \text{H}_2\text{O}$), but is a different substance, which according to the molecular proportions of the chemical components, as shown below, may be called zinc meta-silicate (ZnSiO_3 or $\text{ZnO} \cdot \text{SiO}_2$). The following is the analysis of the precipitate Y:

	Grams	Per Cent.	Molecular Weight	Molecular Ratio	Molecular Proportion
SiO_2	1.0120	42.600	60.4	0.705300	1
H_2O	0.0003	0.012	18.0	0.000667	0
ZnO^*	1.3610	57.280	81.4	0.703800	1
	2.3733	99.892			

$$0.7038 : 0.7053 = 1 : 1 = \text{ZnO} : \text{SiO}_2 = \text{ZnSiO}_3.$$

The filtrate from the precipitate Y was colloidal. It was heated to boiling for half an hour until CO_2 was no longer given off. During the heating, a precipitate collected as expected. After being cooled, the precipitate was removed by filtration from the colloidal solution and was washed with distilled water. It was dried in the oven and cooled in the desiccator, and weighed 1.9747 g. For clearness, it will be designated as precipitate Z, from which were obtained:

	Grams	Per Cent.	Molecular Weight	Molecular Ratio	Molecular Proportion
SiO_2	0.4634	23.470	60.4	0.3886	1
H_2O	0.1381	6.993	18.0	0.3885	1
ZnO	1.2490	63.240	81.4	0.7769	2
A gelatinous residue	0.1235	6.253			
	1.9740	99.956			

$$0.7769 : 0.3886 : 0.3885 = 2 : 1 : 1 = 2\text{ZnO} \cdot \text{SiO}_2 \cdot \text{H}_2\text{O}.$$

From the molecular proportions of X, it can be readily seen that the residue consists of silicate and carbonate of zinc, as the undissolved portions of their original amount, or the proportion of the oxide of zinc is almost equal to the sum of those of the other three components, namely

* Zn in this case was determined as pyrophosphate and ZnO was calculated from the molecular formula of $\text{Zn}_2\text{P}_2\text{O}_7$ and found to contain 1.082 grams metallic zinc.

$0.2244 + 0.3656 + 0.2241 = 0.8145 - 4$ (to be exact) or the amount of SiO_2 , CO_2 , and H_2O = the amount of ZnO .

Thus $3\text{ZnO} = \text{SiO}_2 + \text{CO}_2 + \text{H}_2\text{O}$ or



On the basis of the original amounts used at the beginning of the experiment and by means of their molecular proportions, it is found that the residue X constitutes 2.151 g. or 53.966 per cent. of hydrous silicate of zinc and 1.82 g. or 45.5 per cent. of the normal carbonate of zinc as representing the undissolved portions from their original amounts; that from the precipitate (Z) were recovered 1.851 g. or 46.27 per cent. of the original hydrous silicate of zinc. In other words, no change has taken place in the hydrous silicate of zinc, so far as its amount and chemical combination are concerned, whereas the amount of zinc carbonate (its original weight is 4 g.) left in the residue X is only 1.82 g. or 45.5 per cent. of the original amount. Its remainder (theoretically 2.18 g. or 54.5 per cent.) has undoubtedly contributed to the precipitation of the meta-silicate of zinc, the precipitate Y, as will be shown in the following:

	Grams
The original weight of ZnCO_3	4.0000
The amount left in residue X.....	1.8200
The amount lost.....	2.1800

According to its molecular weight, 2.18 g. ZnCO_3 contain 1.137 g. metallic zinc, and if this theoretical amount had all been precipitated out as zinc meta-silicate, there would be 2.464 g. ZnSiO_3 as against the actual value of 2.376 g. ($2.3763 - 0.0003 = 2.376$). The difference between the two is about 3.5 per cent., which is evidently lost in the course of manipulation.

From these results, it is clear that zinc carbonate may be replaced by a solution of soluble silicate to form zinc silicate, but empirically, this compound in no way resembles calamine, since the water is not present in sufficient amount. The water, in the case of true calamine, would constitute one-half of the molecular proportion of the oxide of zinc as required by the formula, $2\text{ZnO} \cdot \text{SiO}_2 \cdot \text{H}_2\text{O}$.

It is also, however, a recognized fact that the water contained in calamine is basic and the mineral is probably a basic meta-silicate of zinc. For this reason, its formula may be represented by $\text{Zn}(\text{OH})_2 \cdot \text{ZnSiO}_3$ in place of $2\text{ZnO} \cdot \text{SiO}_2 \cdot \text{H}_2\text{O}$. (See the experiment under 6.)

6. *The Reactions of a Mixture of Hydroxide and Silicate of Sodium with a Dilute Solution of Zinc Sulphate.*—From the above suggestions it may be readily inferred that a mixture of Na_2SiO_3 and NaOH will precipitate some such compound, partly as zinc hydroxide, and partly as meta-silicate, from a dilute solution of a zinc salt. The following experiment

was designed in an attempt to demonstrate these suggestive properties of the mineral under discussion.

Experiment 11.—2,000 c.c. of very dilute solution of ZnSO_4 (1.61 g. ZnSO_4 per liter) were treated with an equally dilute mixture of Na_2SiO_3 and NaOH in excess. A rather heavy flocculent precipitate gradually collected. The amount obtained was 2.412 g. and yielded:

Chemical Components Per Cent.		Mineral Components Per Cent.	
H_2O	7.463	ZnSiO_3	58.83
SiO_2	25.020	$\text{Zn}(\text{OH})_2$	41.21
Zn	54.180 = 67.47 ZnO		
	<hr/> 99.953		<hr/> 99.95

According to this analysis, the amount of zinc in the precipitate was 1.307 g., as against 1.308 g. originally present in 2,000 c.c. of ZnSO_4 solution. By recasting from its chemical components, it is found that there are 1.4174 g. as zinc meta-silicate (ZnSiO_3) and 0.994 g. as zinc hydroxide or 58.83 per cent. ZnSiO_3 and 41.21 per cent. $\text{Zn}(\text{OH})_2$. If we consider the precipitate obtained in this way as a single compound, the writer sees no objection to writing $\text{ZnSiO}_3 \cdot \text{Zn}(\text{OH})_2$ for the chemical composition of calamine as a makeup of one part hydroxide and one part meta-silicate, since neither of the formulæ, $2\text{ZnO} \cdot \text{SiO}_2 \cdot \text{H}_2\text{O}$ and $(\text{ZnOH})_2 \cdot \text{SiO}_3$, given in Dana's *Mineralogy*, has any greater claims to being representative. It is far from safe, however, to regard this amorphous precipitate as true calamine, for the reason that, with the exception of its index of refraction (very close to 1.62), no optical confirmation could be obtained from it.

In spite of this lack of sufficient optical evidence, however, there is reason to believe that calamine in nature is probably built up by steps as meta-silicate and hydroxide in their incipient stages, through some natural influence or condition, producing a continuous alternation in the formation of this mineral, which we are not able to imitate in the chemical laboratory. Since even a very dilute solution of soluble silicate (Na_2SiO_3) and zinc salts always gives an amorphous precipitate at the ordinary temperature and pressure, and since amorphous calamine is not encountered in nature, the writer is constrained to believe that the natural calamine must have crystallized very slowly from very dilute siliceous solutions.

D. High Temperature Methods

7a. *The Influence of Temperature on the Formation of Calamine.*—*Experiment 12:* In connection with the preparation³⁵ of willemite accord-

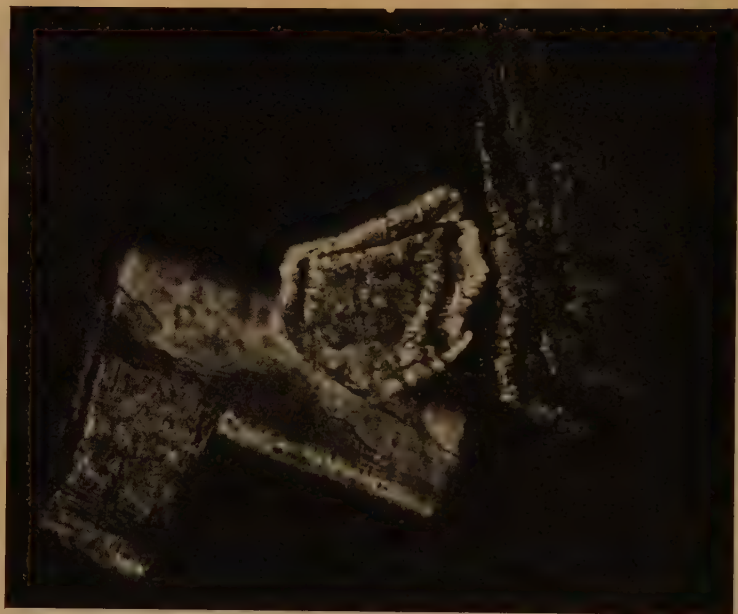
³⁵ The writer desires to thank R. J. Moore of the Department of Chemistry at Columbia University most heartily for placing at his disposal the apparatus without which it would have been impossible to conduct satisfactorily any of these experiments, and C. J. Morrell for kind assistance.

ing to M. Gorgeu's method,³⁶ the writer succeeded in obtaining minute crystals of calamine instead, by treating a mixture containing one part of H_4SiO_4 and 30 parts of one equivalent of Na_2SO_4 and one-half equivalent of ZnSO_4 . In these experiments only anhydrous Na_2SO_4 and ZnSO_4 were used. Various amounts (from one-half to one equivalent) of the latter compound in different experiments were tried, but one-half equivalent of ZnSO_4 with one equivalent of Na_2SO_4 in 30 parts of these two compounds, is the suitable amount. All three compounds, well mixed in a porcelain crucible made for high-temperature work, were first heated over a Bunsen burner; and when the fusion began, the crucible was transferred to the "asbestos nest" built in an iron vessel, which was placed in the middle of an assay furnace previously heated. By so doing the temperature inside of the "asbestos nest" may be easily kept at or below 900°C . (this was measured by a LeChatelier thermo-couple pyrometer, protected by a long porcelain tube, which was inserted vertically into the "nest") in order to prevent any zinc from volatilization during the fusing, while the temperature outside the iron vessel is probably over $1,000^\circ\text{C}$. The heating in this was continued from one-half to one and one-half hours. If the porcelain crucible is transferred into the furnace without using the "asbestos nest," the results are comparatively poor and indicate the volatilization of zinc by the sudden increase of temperature and quick fusion. In this case the odor of SO_2 is quite noticeable near the furnace. On the other hand, it has been experimentally shown that, when the fusion is quite complete, if the "asbestos nest" is used, the temperature may be raised to $1,300^\circ\text{C}$. or $1,400^\circ\text{C}$. without noticeable volatilization of zinc.

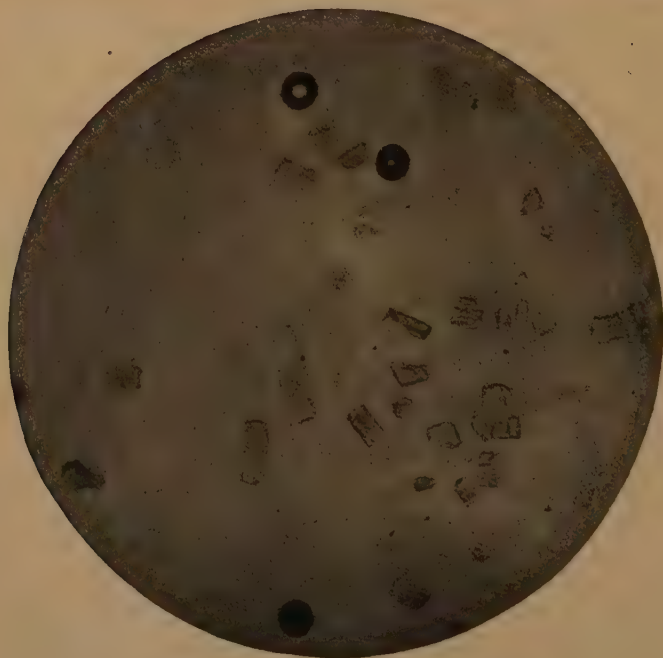
After the melt had been kept in the furnace for half an hour or more, the fused mass, which usually became covered with a thin yellowish and sometimes greenish sheet, was placed directly in boiling water. This boiling water treatment causes solidification and usually gives a green surface to the solid mass, especially along the edge. If the mass has been heated too long in the furnace, the solution from the boiling water usually shows a basic reaction; if under-heated the reaction with litmus paper is neutral. By heating in the furnace from half to one hour in the "asbestos nest" as described above, we secure the best results.

After the fused mass, solidified in the boiling water, had been cooled, the contents were taken out with a knife and boiled in distilled water for a long time in order to remove any sulphates crystallized out on the sudden cooling. The desired silicate, if formed, would remain insoluble either in masses or in crystals. This is found to be true, but in each case the crystals of the silicate, as proved below, are very minute and are so small in amount that it is difficult to separate them from the greenish

³⁶ *Bulletin 10, Société, Française de Minéralogie*, pp. 30 to 39.



No. 1.



No. 2.

FIG. 4.—MICROPHOTOGRAPHS OF THE CRUSHED FRAGMENTS FROM THE SYNTHETIC CRYSTALS OF CALAMINE PRODUCED BY HEAT FROM A CHEMICAL MIXTURE.

No. 1, magnified 215 diameters, crossed nicols. No. 2, magnified 25 diameters, plain light.

fine residues which equally remain insoluble and which, by qualitative tests, gave SiO_2 , Zn, and water vapor.

These minute crystals of the silicate are different in shape and may be divided into two kinds, respectively white and light green in color. The white ones are more or less tabular in form and fibrous in structure, while the green ones seldom show any definite crystal outline. There is no difference in their chemical composition, but optically the white silicate is positive, whereas the green is negative though both give biaxial interference figures.

As suggested by Mr. Colony, Professor of Mineralogy and Geology in Cooper Union in the City of New York, the white crystals which are optically positive and biaxial may be regarded as *m* calamine; the green ones which are optically negative and biaxial, may be considered as *n* calamine. The latter probably represents a transitional stage in the formation of calamine proper. It is further thought that such variation is not uncommon in the behavior of many minerals. Experience shows that a mineral may under special conditions, change even its system as in case of leucite³⁷ which cannot remain isometric below 560° C.

The following are the results obtained from the microscopic examinations (see Fig. 4):

	White Crystal	Green Crystal
Shape.....	Elongated	Lath shaped and elongated
Color (transmitted light)	Light bluish	Light greenish
Indices of refraction.....	1.63	1.61
Extinction.....	Parallel	Parallel
Extinction angles.....	85°-90°	Mostly 90°
Birefringence.....	<0.04	>0.06
Elongation.....	+	-
Interference figure.....	Biaxial	Biaxial
Optical sign.....	+	-

The following are the chemical analyses of the respective silicates:

	White			Green			
	Per Cent.	Molecular Weight	Molecular Ratio		Per Cent.	Molecular Weight	Molecular Ratio
H ₂ O	7.457	18.0	0.4143	H ₂ O	6.825	18.0	0.3792
SiO ₂	24.990	60.4	0.4137	SiO ₂	25.030	60.4	0.4144
ZnO	67.480	81.4	0.8289	ZnO	67.503	81.4	0.8292

The molecular proportion of water for the green silicate is about 0.82 per cent. lower than what is theoretically required, but this difference

³⁷ N. H. and A. N. Winchell. *Elements of Optical Mineralogy*, p. 275 (1909).

is so small that it may be safely attributed to the manipulation. Aside from this, either silicate has the following relation, viz. $\text{H}_2\text{O} = 1$, $\text{SiO}_2 = 1$, $\text{ZnO} = 2$. In other words these silicates or crystals have the formula $2\text{ZnO} \cdot \text{SiO}_2 \cdot \text{H}_2\text{O}$, or $(\text{ZnOH})_2\text{SiO}_3$ or $\text{Zn}(\text{OH})_2 \cdot \text{ZnSiO}_3$ (as suggested by the writer in an earlier paragraph). The question, which is the correct formula, remains to be determined, and is hardly within the province of this work.

From these chemical and microscopic results it is clear that the crystals obtained under the conditions described above, are hydrous silicate of

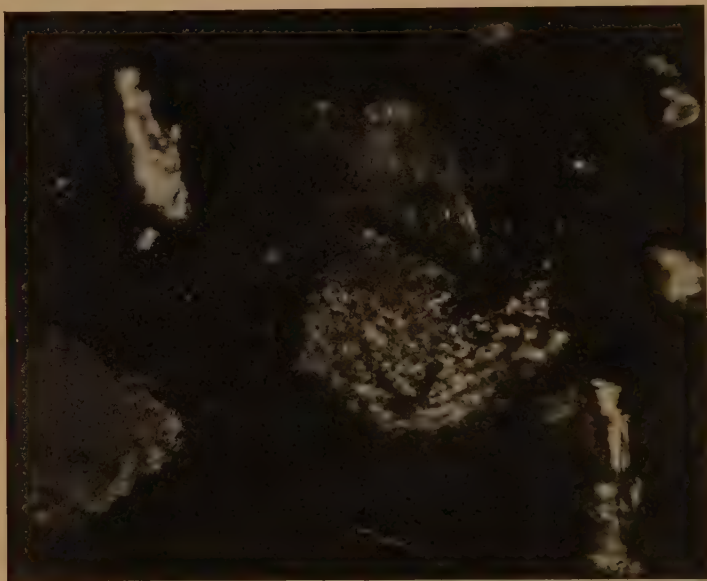


FIG. 5.—MICROPHOTOGRAPH OF THE CRUSHED FRAGMENTS FROM THE FUSED NATURAL CALAMINE.

These are some of the fragments showing the tendency of rearranging their molecules. Under high power some of the very minute hexagonal crystals in the fragments are well shown. This is no doubt due to the influence of heat, but these newly rearranged crystals are too minute to show on photograph. Magnified 215 diameters. Crossed nicols.

zinc or mineralogically a calamine instead of M. Gorgeu's willemite which is an anhydrous silicate of the same metal. The water content³⁸ of the crystals, under our chemical examination, did not pass off until heated above the Meeker burner with a strong flame. From these experiments the writer concludes: (1) that M. Gorgeu probably mistook the hydrous silicate for the anhydrous silicate of zinc by neglecting its water content (of which he made no mention in the chemical analysis of his product,

³⁸ According to Fock, "the water goes off only at a red heat, the mineral being unchanged at 340°C . Dana. *System of Mineralogy*, Sixth Edition, p. 548.

and his optical description is rather insufficient); and (2) that he probably took natural calamine instead of willemite (in some instances these two minerals resemble each other and are difficult to distinguish, especially when they are wholly white, and exhibit similar radial structure—as indeed is rare, yet occurs in some of the white willemite from Franklin Furnace, N. J.) for comparison with his synthetic product, since he declared that the elongation of his crystals was parallel to c —which is one of the common characteristics of most of the fragments from the crystals obtained by the writer, and which is very common with natural calamine as well as willemite in thin sections. It, therefore, cannot be used as a criterion for the determination of either mineral.

In one respect this synthetic work on calamine may seem to some of the geologists to be inadequate, namely the temperature and the boiling-water treatment of formation. Field observation points to ordinary temperature. Doubtless, it would be highly desirable to obtain similar crystals of calamine at ordinary temperatures. It must be remembered, however, as pointed out some time ago by well-known geo-chemists, “that there is nothing in our knowledge of the minerals to indicate anything more than continuous changes in the necessary conditions as the temperature falls.” It must also be admitted that to imitate a process in nature will be always more or less a matter of extrapolation.

7b. Possible Effect of Heat on Calamine and Smithsonite.—Experiment 13.—It is known that artificial zincite and willemite occur in furnace slags and in iron and lead furnaces from the ores carrying zinc. It has been suggested by Dr. Kemp that the ores of zinc at Franklin Furnace and Ogdensburg, N. J., are “the result of metamorphism.” This suggestion led the writer to investigate the effect of heat on calamine and smithsonite. The experiments described below are of very simple character, consisting of only four steps:

1. Fusion of the Natural Specimens of Calamine and Smithsonite.—Mere heating in a good gas furnace does not produce any result beyond driving off H_2O from calamine and CO_2 from smithsonite. The resultant products are usually crumbly and brittle, although they still retain the original structures. No optical change was observed. It is worthy of note, however, that when water is driven off from calamine by heating, the product shows the fluorescent property which is commonly used in testing willemite at Franklin Furnace, N. J. The writer obtained the fluorescence by burning “magnesium ribbons” before the sample.

The results of fusion are quite different. The H-O flame was employed for the fusion, and the fused mass was cooled slowly in each case in the neighborhood of the flame. Fragments of the fused portions of the respective minerals were mounted on slides.

2. Examination under the Microscope.—The fused fragments from smithsonite gave uniaxial positive properties indicating a change in op-

tical sign, since for smithsonite, the optical sign is negative. (Out of hundreds of fragments only a few showed this change the rest were overfused and became cloudy under the microscope.) With the fused calamine, some difficulties were experienced, but the writer succeeded in finding some fragments showing uniaxial positive properties, indicating that the mineral had changed its interference figure; since for calamine, the interference figure is biaxial. It was also observed under the microscope in some fragments that the calamine was completely changed being fully composed of minute six-sided crystals. This seems to indicate the molecular change by recrystallization to willemite.

3. Chemical Analysis.—These analyses do not demonstrate that the fused specimens are molecularly changed; they merely indicate the H_2O and CO_2 are absent after the heat treatment.

	Fused Calamine				Fused Smithsonite	
	Per Cent.	Molecular Weight	Molecular Ratio	Molecular Proportion		Per Cent.
SiO_2	27.06	60.4	0.4584	1	ZnO	100.18
ZnO	72.93	81.4	0.8960	2	CO_2	None
H_2O	None					

4. Test for Fluorescence.—The fused portion from calamine shows fluorescent color, though less strongly than the willemite from Franklin Furnace, N. J.

These results strongly suggest that under the application of both heat and pressure, as they commonly exercised their influence under metamorphic conditions, still more striking and definite changes might be exhibited. But this cannot be obtained without the necessary equipment for both pressure and temperature. A system of slow cooling is also essential.

E. Summary of the Experimental Work

The oxidation and solution of blende is prerequisite to the formation of any one of the oxidized ores, and the process of oxidation is brought about by reactions of the sulphide with a ferric iron salt. Ferric sulphate is most conveniently used.

When the iron salt is ferric sulphate the zinc sulphide is usually converted into sulphate; but the solvent action of this salt does not seem to be in the proportion to its concentration, as only a small percentage of the ferric is reduced to ferrous by the oxidation of the zinc sulphide. These results suggest that the chemical equation, $\text{Fe}_2(\text{SO}_4)_3 + \text{ZnS} = 2\text{FeSO}_4 + \text{ZnSO}_4 + \text{S}$, represents only the chemical reaction and not the chemical ratio.

When a ferrous salt is alone present (as it rarely is, in quantity, in the zone of oxidation), no oxidation of zinc sulphide can take place unless free air is accessible; and when air is introduced, the sulphide may be converted into sulphate by the oxidation products of the ferrous salt prior to the formation of any one of the oxidized ores of zinc.

Soluble salts of such metals as Ag, Cu, and Pb in solution, exert a powerful solvent action on blende. Of these, a solution of silver salt is much more powerful than a copper salt solution and the latter is more so than a salt solution of lead. Lower and different concentrations of the sulphates of silver and copper were used. In case of lead, lead nitrate was substituted.

A solution of cupric sulphate has a powerful solvent action on the natural sulphide of zinc, hence in an orebody containing much copper sulphide or galena or both, little sphalerite may be expected in the horizons where the former two sulphides are rich, as most of the blende will be taken into solution and the metal will be redeposited somewhere as one of the oxidation products; and conversely, if rich zinc sulphide be found in a certain horizon, we may be pretty sure that the orebody cannot have contained much lead (or copper) sulphide, and that little oxidation products should be expected. This accords with the fact that zinc carbonate and zinc sulphide have been found in different horizons, as if they had no relation to each other, as in the Circle Mines at Oronogo, Mo.

When a solution of an alkali or calcium bicarbonate is mixed with a solution of zinc sulphate or chloride, a zinc carbonate is precipitated; but when a reagent containing normal carbonate, such as Na_2CO_3 or K_2CO_3 is used, only a basic carbonate or hydrozincite is obtained.

When a dilute silicate solution (Na_2SiO_3) is mixed with a dilute solution of zinc salt, only a precipitate of a meta-silicate of zinc is obtained. In the presence of carbon dioxide, this silicate solution replaces also zinc carbonate, to form meta-silicate of zinc. If the reagent contains a mixture of sodium silicate and sodium hydroxide a "complex" precipitate of meta-silicate and hydroxide of zinc may be obtained. In chemical composition this precipitate corresponds to natural calamine, but by reason of its amorphous character no definite confirmation is here offered.

The presence of carbon dioxide exerts a powerful solvent action on all the oxidized ores under consideration. Of these, zinc carbonates are more soluble than the basic silicate of zinc in waters containing an excess of carbon dioxide. This may account for the fact that no carbonic acid, as far as we know, has ever been detected in the analysis of the waters from the mines of these oxidized ores. Upon release of the carbon dioxide any of these ores may be recovered.

Limestones immersed in dilute solutions of zinc sulphate were found to be replaced by zinc carbonate; but the replacement is much more rapid with the impure limestone than with the pure ones. This result suggests

that the presence of organic matter in the impure limestone has probably exerted some influence in the rate of replacement.

At or a little below 900° C., minute crystals of calamine may be prepared from a chemical mixture containing definite proportions of H_4SiO_4 , Na_2SO_4 (dried), and ZnSO_4 (dried). Both the chemical analysis and the optical characters of the crushed fragments all point to their identity with natural calamine.

The influence of heat on calamine and smithsonite results in the change of their optical characters into those of willemite and zincite; but the optical tests were so difficult to carry out satisfactorily that it appears doubtful whether the anhydrous silicate and the oxide of zinc can be formed in this way.

III. PETROGRAPHIC WORK

Introductory

Petrographic data concerning the mode of formation of the ores of zinc are, so far as we are aware, limited to descriptions of occurrences of blende.³⁹ As to the oxidized ores practically no attempt⁴⁰ has heretofore been made to discuss the paragenesis of the mode of their formation in the light of their petrology. They have naturally been regarded as products from the alteration of sphalerite, but the conclusion was wholly drawn from field studies.

The data given below are the results of a study of thin sections from ore specimens, chiefly selected from the collection of ores of the Department of Geology at Columbia University, and representing only a few localities chosen for this purpose.

This was undertaken in conjunction with the experimental work, in order to arrive at some definite conclusion as to (a) the genetic order of the precipitation of zinc carbonate and zinc hydrous silicate; and (b) the influence of inclosing rocks and associated minerals, principally limestone and pyrite respectively, upon sphalerite and its oxidation products. For this reason the descriptions are not necessarily confined to the oxidized ores alone but also include the sulphide; and the specimens from which thin sections were cut are grouped, not according to their localities but according to their present form of occurrence, into two types as follows: (1) Sphalerite occurring in limestones or cherts from southwest Wisconsin; southwest Missouri; Mascot Mine, Tennessee; northwest Arkansas; Wythe County, Virginia; and Saucon Valley, Pennsylvania; and (2) partially altered sphalerite and the oxidation

³⁹ C. E. Siebenthal. *Mineral Resources*, 1911, p. 353 et seq., U.S. Geological Survey, W. Lindgren, *Mineral Deposits*, p. 422 (1913); and T. L. Watson, *Bulletin No. 1*; *Geological Survey of Va.*, pp. 95 to 96 (1905).

⁴⁰ Watson. *Idem*, p. 45.

products, calamine and smithsonite, from southwest Missouri, northern Arkansas; and Leadville, Colo.

1. *The Occurrence of Sphalerite in Limestones or Cherts*

Different views have been advanced by numerous authors as to the mode of deposition of zinc sulphide in limestones and cherts or in their breccias. Later writers have generally agreed, however, that the ores of this type were first derived from above by the decomposition and erosion of superimposed strata, and finally concentrated by meteoric waters in brecciated limestones or cherts.

The result of the present investigation on thin sections from the sulphide ores of zinc points to some such theory. The sections examined show that in many instances the zinc sulphide has filled completely the interstitial space in the brecciated country rocks, principally limestones and cherts. In others, the sulphide appears to have impregnated the country rocks and filled the cavities in the limestones as if it had been deposited with the inclosing rock.

Among the associated minerals are galena, pyrite, magnetite, limonite, secondary quartz, and calcite, fluorite and greenockite. The latter two were not observed in the sections here examined; and galena was absent, or occurred only in minor amount, where sphalerite was dominant. Sphalerite usually incloses galena, indicating the previous deposition of the latter. Of the three iron compounds, pyrite is the commonest, while magnetite infrequently concentrated around the peripheries of sphalerite and limonite occurs sparingly along cleavage cracks, when such occur. The order of origin of these minerals with respect to the sulphide ore of zinc has therefore appeared to be: (1) Galena (if present); (2) sphalerite; (3) iron sulphides; and (4) secondary quartz and calcite. (2) and (3) may have taken place almost simultaneously and (3) and (4) reverse in some instances. Wherever quartz occurs in these sections, it is invariably of secondary origin.

A specimen from southwest Wisconsin consisted of sphalerite and a siliceous mass on the top, which was thought to be the alteration product, calamine, but upon examination under the microscope was found to consist of aggregates of secondary quartz. Some are cryptocrystalline and others are much larger. In it secondary calcite occasionally occurs (Fig. 6).

The essential relation of secondary quartz to sphalerite appears to be the same in all the specimens studied from the localities indicated above, as shown in Figs. 6 to 11. Irrespective of their localities, they all show that the voids or interstices in the limestone were completely filled with secondary quartz, and that where spaces in the country rock were not wholly occupied by sphalerite the remainder was filled with quartz.

Secondary calcite is occasionally seen in cleavage cracks of sphalerite. These relations indicate that the precipitation and concentration of sphalerite preceded the deposition of secondary quartz and calcite.

Some sections show pyrite penetrating the blende, as well as the calcite and quartz, along their strain- and cleavage-cracks. This means that subsequent to the deposition of most of the sphalerite and of the secondary quartz in the fractured limestone, there came a period of infiltration of solutions, bearing iron sulphide. In these cleavage cracks of sphalerite and quartz, pyrite has been deposited. It further favors the contact of sphalerite and is especially prominent around the edges of the zinc sulphide. The contacts of pyrite and sphalerite thus produced afford the most important factor in the oxidation of sphalerite, discussed in the earlier part of this paper.

Sphalerite is, as a rule, most abundant in brecciated limestone and in the secondary black chert. The latter is largely made up of cryptocrystalline quartz with grains of blende and crystals of dolomite (Fig. 7). The dark color of the chert is due to the presence of much organic matter. In the bedded limestones and white cherts, sphalerite is practically absent.

Summary of the Blende in Limestones and in Cherts.—From the thin sections examined here it may be briefly stated that the deposition of secondary quartz was subsequent to the crystallization of sphalerite. This secondary quartz was probably in part introduced by percolating waters and in part was furnished by the decomposition of cherty limestone and cherts with minor amounts of secondary calcite. The precipitation of pyrite in some cases was almost simultaneous with the crystallization of sphalerite, but it has not had the opportunity since its deposition to exert influence on the oxidation of sphalerite.

2. *The Partially Altered Sphalerite and the Oxidation Products*

The country rock in which the ores of this second type occur is chiefly dolomitic limestone. The associated minerals are galena, pyrite, limonite, hematite (both derived from pyrite), secondary quartz and calcite, and some primary quartz. In no case are they all present in a single section; and primary quartz and pyrite were observed only in specimens from Joplin district, Missouri, and Silver Hollow Mine, Arkansas.

a. *Smithsonite with Blende* (Fig. 18).—This shows the peculiar occurrence of galena around peripheries of residual grains of sphalerite. In some cases galena forms a succession of rings. The sphalerite within a single ring is usually unaffected. The iron oxide associated with smithsonite is usually in the form of limonite. It penetrated or concentrated along the cleavage cracks of smithsonite, as shown in Fig. 18. A characteristic feature of the smithsonite is the occurrence of minute secondary

quartz and of some straight and elongated crystals of calamine intergrown in it, as shown in Fig. 14. It is evident that the intergrowth or interlocking of these oxidation products (Figs. 13 and 14) represents the overlapping periods of formation of both minerals. The amount of each necessarily depended upon the amount of the required substances in the reagents in original solutions.

The ores from Silver Hollow Mine, Arkansas, contain a large number of almost round grains of quartz, some of which were completely surrounded by pyrite and others by both pyrite and secondary calcite. These quartz grains are primary in origin, and are believed to have come from the disintegration of some original sandstone, before the penetration of pyrite. The edges of these quartz grains are more or less corroded and in some cases secondary calcite has filled these spots (Fig. 15). The optical examination of this ore indicates that smithsonite is rarely derived direct from sphalerite, but mostly from solutions, after the sphalerite had been oxidized into a soluble salt presumably a sulphate, by meteoric waters. Fig. 16, from the same section, shows sphalerite cut by a veinlet of smithsonite. In some places the section also shows the sharp contacts of smithsonite with sphalerite. Had the smithsonite been formed by direct alteration from sphalerite such a relation could hardly exist. These sections further indicate that simultaneous with or subsequent to the solution of sphalerite in some parts of the ore there occurred a fracturing of the unaltered sphalerite, which permitted the infiltration of solutions bearing smithsonite. In the cracks thus produced in the unaltered portions of sphalerite, smithsonite has been deposited. Subsequent to its deposition is the deposition of secondary quartz which filled the rest of the vein. Pyrite which penetrated sphalerite and calcite along their cleavage weakness in the specimen is only observed in part in thin sections.

b. *Calamine with Blende*.—Where blende was not entirely altered, its periphery was surrounded by a ring of galena (see Fig. 18) and occasionally by limonite. Like smithsonite, calamine rarely replaced sphalerite in place. Its crystals usually exhibit a habit of radial or of massive structure. This habit of structure (Figs. 13 and 17) suggests at least one thing, namely, that its crystallization in general took place from solutions, and not by direct alteration of sphalerite.

The rich ore of calamine may also carry some smithsonite as a minor product, and the same is also true in the reverse case. Both of them may replace calcite as the result of metasomatic replacement, as shown on Fig. 18. It is sometimes difficult to distinguish smithsonite from calcite, especially when they are closely associated. The difference in their relief generally affords a clue to their distinction. In nearly all cases, the surface of smithsonite appears very shaggy under the microscope.

c. *Smithsonite and Calamine*.—The sections here examined were cut

from the ores containing both calamine and smithsonite and are practically free from gangue. In some instances they are intergrown in each other; in others they are interlocked with each other, as already described above.

As to their structure, each of these two minerals seems to have followed a definite habit of crystallization. With calamine the radial structure is common, whereas smithsonite is usually granular, especially in high-grade ore (Figs. 13, 14, 17).

A specimen, known to contain both calamine and smithsonite, from Leadville, Colo., was found to be mostly calamine, largely made up of cryptocrystalline and elongated crystals, as shown in Fig. 19.

Fig. 20 from the same section shows narrow strips of iron oxides (hematite and limonite) surrounded by calamine. These oxides appear to have concentrated along the partings and the cleavage cracks of calamine. Where this mineral is in the form of minute crystals, the surface is irregularly coated also with iron oxides (Fig. 19), chiefly limonite. The iron oxide was undoubtedly an iron sulphide in origin, and was later reconcentrated and reprecipitated in its present form, after its oxidation product, generally ferric sulphate, had been destroyed in the oxidation of sphalerite, with which it was in contact in the original rock.⁴¹

Summary on the Partially Altered Sphalerite and its Oxidation Products.
—From the examination of these sections it may be said that:

(a) When sphalerite was surrounded by a ring of galena or iron oxide, its further oxidation was terminated.

(b) Sphalerite was rarely replaced by its oxidation products in place. In most cases the solution of sphalerite, through the reaction of the oxidation products of iron sulphides, preceded the transportation or migration, and the deposition of the metal in the form of carbonate or hydrous-silicate.

(c) The alteration of sphalerite proceeded inward zonally from the exterior.

(d) No definite order of the formation of calamine and smithsonite was observed. They may intergrow with each other or either may precede the other. Thus the formation of these oxidation products depended upon the availability of the required substances in the reagents which had come into contact with a solution containing the metal zinc, presumably as zinc sulphate.

(e) Both calamine and smithsonite replaced calcite, or the limestone might pass into these products in orebodies. This relation has also been observed in some zinc mines.⁴²

⁴¹ T. L. Watson. *Bulletin No. 1, Geological Survey of Va.*, p. 45 (1905).

⁴² U. S. Grant. Report on the Lead and Zinc Deposits of Wisconsin, *Bulletin No. 14, Wisconsin Geological and Natural History Survey*, p. 59 (1906).

IV. CONCLUSIONS

From the chemical and petrographic data presented in this paper we have sufficient evidence to justify the following conclusions:

1. Sphalerite is first attacked by the oxidation products of iron sulphides before the formation of the secondary minerals containing zinc.

2. The formation of the oxidized ores rarely occurs by direct replacement of sphalerite.

3. Normal zinc carbonate (smithsonite) is formed when the reagents in solutions are bicarbonates; and basic zinc carbonate (hydrozincite) is formed when the reagents are normal carbonates.

4. Limestones may be replaced in nature by solutions of a zinc salt, usually the sulphate.

5. In nature both smithsonite and calamine replace limestones.

6. No free carbon dioxide can be present where silicates and carbonates of zinc occur, and conversely, these minerals of zinc do not appear where free carbon dioxide is present.

7. "Amorphous calamine" may be formed from solutions of silicate and hydroxide under ordinary conditions.

8. Calamine in nature is probably formed by steps, and is precipitated from waters containing ions of some gelatinous silicate and of hydroxides.

9. It is evident that calamine is formed under ordinary conditions in nature, but time is a great factor in the process. Until this factor is satisfied we can hardly expect the production of crystals of calamine by wet methods.

10. Minute crystals of calamine may be formed by heat at or below 900°C. from a chemical mixture, as set forth under Experiment 12.

11. No confirmation is here offered of the change of calamine and smithsonite to willemite and zincite by heat, other than that heat produces changes of optical characters into those of willemite and zincite respectively.

V. MICROPHOTOGRAPHS AND PHOTOGRAPHS WITH THEIR EXPLANATIONS

Abbreviations

C = calamine

Ca = calcite

D = dolomite

G = galena

I = iron oxides

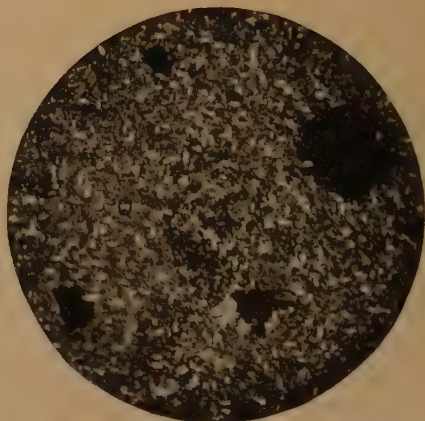
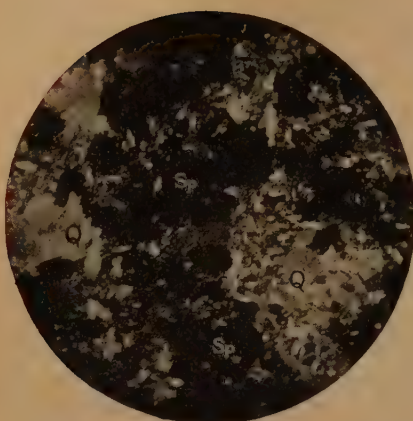
P = pyrite

Q = secondary quartz

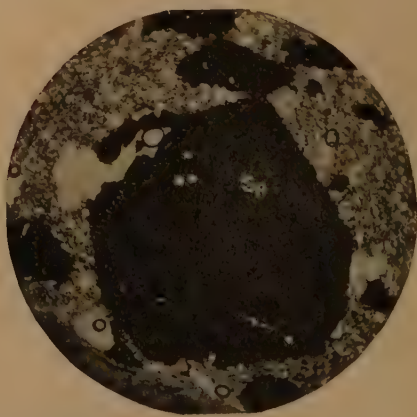
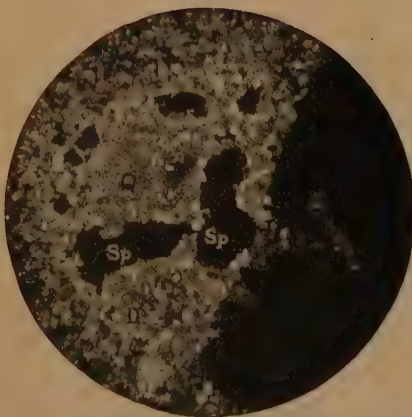
PQ = primary quartz

S = smithsonite

Sp = sphalerite



FIGS. 6 AND 7.—MAGNIFIED 17 DIAMETERS; CROSSED NICOLS.



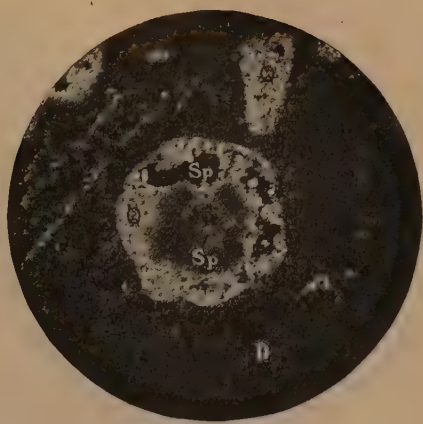
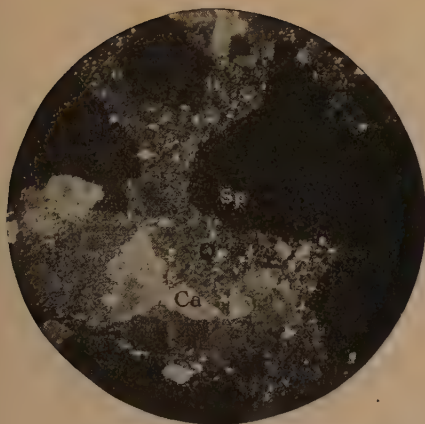
FIGS. 8 AND 9.—MAGNIFIED 17 DIAMETERS; CROSSED NICOLS.

FIG. 6.—Thin section of blende ore from Southwest Wisconsin. Black areas represent sphalerite. White areas, secondary quartz, which was formerly thought to be zinc silicate such as calamine.

FIG. 7.—Thin sections of black chert containing ore from Oronogo Circle Mine, Oronogo, Mo. Black areas, sphalerite. Shows cryptocrystalline character of the secondary quartz in the black chert.

FIG. 8.—From the same locality as 7. Shows the voids in the limestone were entirely filled with both sphalerite and secondary quartz.

FIG. 9.—Thin section of the sulphide ore from Joplin, Mo. Note the sphalerite was entirely surrounded by secondary quartz.



FIGS. 10 AND 11.—MAGNIFIED 17 DIAMETERS; CROSSED NICOLS.

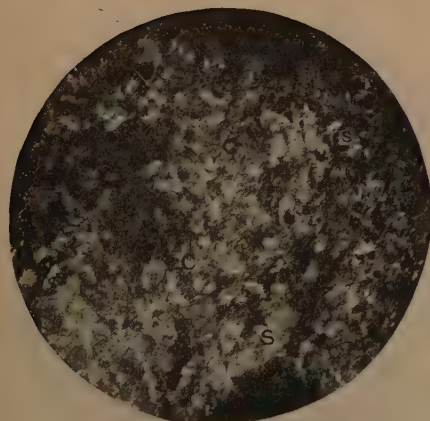


FIG. 12.—MAGNIFIED 17 DIAMETERS;
CROSSED NICOLS.

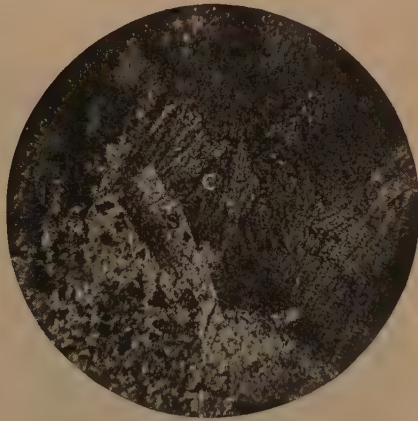


FIG. 13.—MAGNIFIED 140 DIAMETERS;
PLAIN LIGHT.

FIG. 10.—Thin section of the ores from Northwest Arkansas. Shows the secondary quartz and the filling of minute cracks accompanied by replacement of the sulphide by the quartz. Re-entrant angles along some margins of the sulphide are well shown.

FIG. 11.—Thin section of the ores from Mascot Mine, Tenn. The dark area within the lighter circular portion represents sphalerite. A large portion of it was probably removed by a siliceous solution from which the aggregates of secondary quartz deposited around the remnants of the sulphide.

FIG. 12.—Thin section of the oxidized ores from San Francisco Mine, Missouri. Shows the intergrowth of calamine and smithsonite. The latter constitutes the major portion of the section including the dark areas; while the latter is only in minor amount and mostly in elongated form of minute crystals.

FIG. 13.—From the same locality as 12. Shows the growth of smithsonite in calamine. Note the difference in their structure. Calamine exhibits its radial habit of crystallization, whereas smithsonite is in lath shape.



FIG. 14.—MAGNIFIED 17 DIAMETERS;
PLAIN LIGHT.

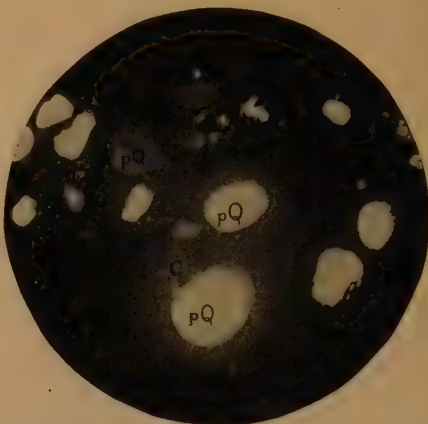
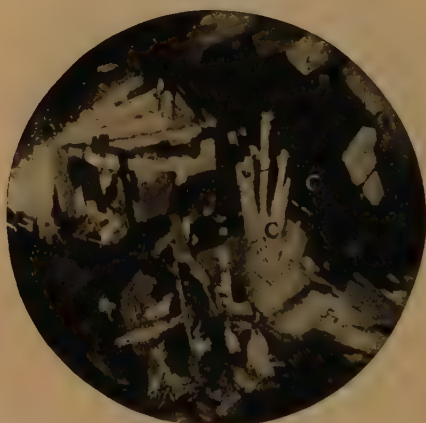
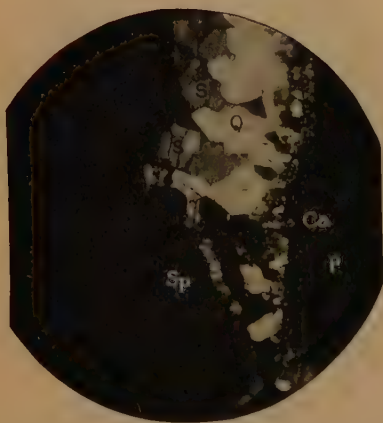


FIG. 15.—MAGNIFIED 17 DIAMETERS;
CROSSED NICOLS.



FIGS. 16 AND 17.—MAGNIFIED 17 DIAMETERS; CROSSED NICOLS.

FIG. 14.—From the same locality as No. 13. Note the interlocking of calamine and smithsonite. The darker surface of calamine is due to the cloudy appearance of the crystals.

FIG. 15.—Section of the ore containing blende and smithsonite from Silver Hollow mine, Arkansas. Note the round grains of quartz and their corroded edges filled with pyrite and secondary calcite.

FIG. 16.—From the same locality as No. 15. Note the veinlet of smithsonite in sphalerite and their sharp contacts. In this, the oxidized ores had probably removed only a short distance and almost remained in places, but it nevertheless indicates the fact that the solution of sphalerite preceded the deposition of smithsonite.

FIG. 17.—Thin section of calamine ore from Joplin, Mo. Note the radial habit of growth in almost all directions.

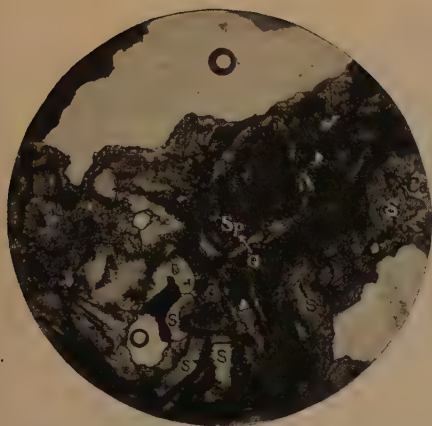


FIG. 18.—MAGNIFIED 17 DIAMETERS;
PLAIN LIGHT.

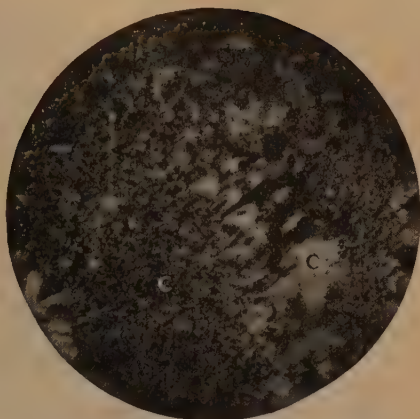


FIG. 19.—MAGNIFIED 17 DIAMETERS;
CROSSED NICOLS.

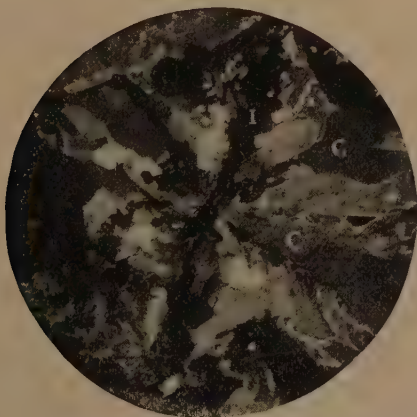


FIG. 20.—MAGNIFIED 17 DIAMETERS; CROSSED NICOLS.

FIG. 18.—Thin section of the ore specimen from Northwest Arkansas. Shows the galena "rings" in dark color inclosing the residual grains of sphalerite. The areas outside the innermost "rings" were replaced by smithsonite. Note also the replacement of calcite by smithsonite to the right. This probably indicates the fact that upon the solution of sphalerite, a carbonate solution immediately came in and gave rise to the precipitation of zinc carbonate. See discussions under the formation of zinc carbonate in the earlier part of this paper.

FIG. 19.—Thin section of the oxidized ores from Leadville, Colo. Note the two types of crystals; one is in an elongated form and the other is mostly cryptocrystalline. Its cryptocrystalline character almost resembles that of some of the secondary quartz in the blende-bearing cherts. (See No. 7)

FIG. 20.—From the same locality as No. 19. Note the narrow strips of iron oxides concentrated along the crystallographic partings and the cleavage cracks of calamine.

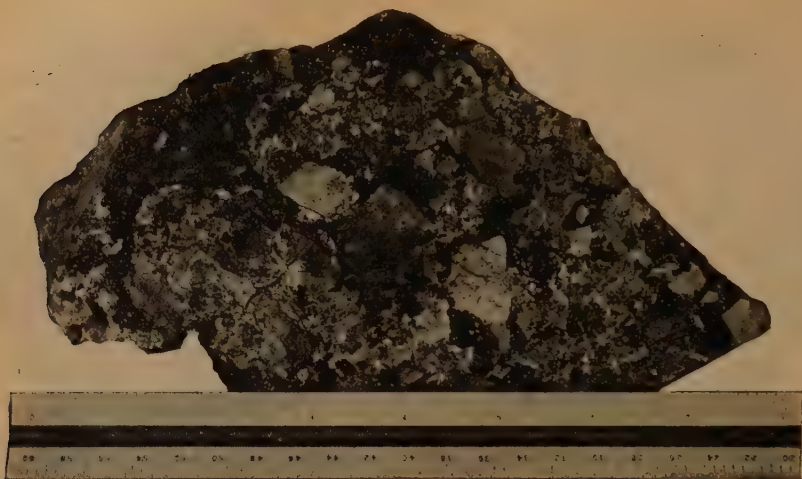


FIG. 21.

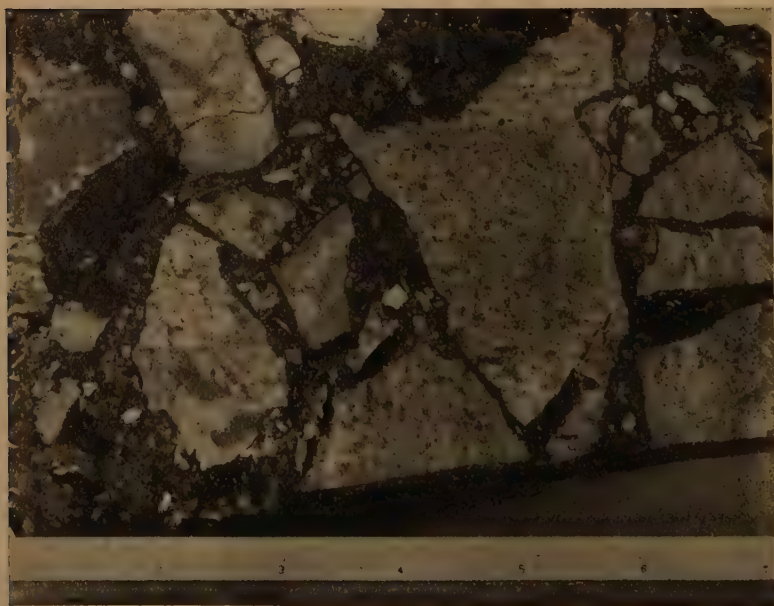


FIG. 22.

FIG. 21.—Specimen of blende ore from Oronogo Circle Mine, Oronogo, Mo. Brecciated structure of the country rock. Dark areas represent sphalerite and the fine grains of black chert cemented together. The lighter portions represent brecciated limestones and white cherts.

FIG. 22.—From the same locality as No. 21. The white areas represent white cherts in which practically no sphalerite occurs.

The Salida Smelter

BY F. D. WEEKS, * NEW YORK, N. Y.

THE Salida smelting plant, owned by the Ohio & Colorado Smelting & Refining Co., is situated at an altitude of 7,000 ft., about 2 miles west of Salida, Colo., and 215 miles southwest of Denver. Salida is a division point on the Denver & Rio Grande R. R. where the narrow-gage line serving the San Juan region joins the through standard gage. This narrow-gage road also serves the coal and coke companies at Crested Butte and the lime quarries and mines at Monarch and Garfield, which are only 20 miles distant. These facts, together with the possibility of an appropriate site, doubtless influenced the builders to locate at this point.

The smelting plant and yard occupy 80 acres of land on the northeast side of the Arkansas River. The slope of the ground is such as to permit of the ore being delivered by cars at a point 25 ft. above the feed floor; and in front of the furnaces there is a fall of 50 ft. for the slag dump. A recent purchase of river bottom has provided slag room for 20 years.

As originally constructed, the plant consisted of four blast furnaces, 40 by 144 in. at tuyères, two copper-matting furnaces, dust chamber, stack, power plant, blowers, sampling works, and one railroad trestle about 16 ft. from bed floor to rails. Later 12 hand roasters of excellent design were added and the practice of smelting leady matte in a copper-matting furnace was abandoned. Since the plant passed under the present management the equipment has been increased so that the following is a brief description of the same at this date: Two lead blast furnaces, 48 by 180 in. at tuyères, 17 ft. 6 in. tuyères to feed floor; four lead blast furnaces, 40 by 144 in. at tuyères, 17 ft. 6 in. tuyères to feed floor; two copper-matting furnaces, not used; blast-furnace flue, 900 ft.; area 245 sq. ft.; blast-furnace stack, 150 ft.; above furnace floor, 190 ft.

When smelting a 500-ton charge per day, the gases approximate 63,700 cu. ft. per minute and travel at a speed of 260 ft. per minute.

The slag is hauled by a 10-ton electric locomotive; the slag shells and settler refuse are elevated to a bin on the feed floor by an electric hoist. The matte is tapped in 1,600-lb. cakes and loaded into cars

* Metallurgical Engineer.

without breaking, in the case of matte low in copper; copper matte is broken by hand before going to the sampler.

The bullion is tapped into pots holding 10 bars and poured into drossing kettles of 30 tons capacity, then drossed, sampled, and shipped. The furnaces, which are fed by hand, are not entirely satisfactory, because the brick walls are so thick that the feeders cannot get close to their work; and further, the furnaces have cast-iron jackets with an end bosh. These faults have not been corrected, for the reason that it would require rebuilding the furnaces, and lead smelters are not at present incurring any expense that can be avoided.

The roasting plant at present consists of three Godfrey furnaces, 26 ft. in diameter, with a capacity of 25 tons per day each; one Wedge furnace, 21 ft. in diameter, with a capacity of 75 tons per day; four Dwight-Lloyd machines, 42 by 264 in., with a capacity of 80 tons per day each. The Godfreys and Wedge are used for preliminary roasting before sintering on the Dwight-Lloyd machines, the sulphur in high-sulphur ores being reduced to a point where the product can be added to the Dwight-Lloyd charge without raising the sulphur above 15 per cent.

These furnaces are served by a 468-ft. flue, having a cross-sectional area of 166 sq. ft., and joined to a stack 85 ft. high, 12 ft. in diameter.

The volume of gases discharged is approximately 125,000 cu. ft. per minute.

The product from the Godfreys and Wedge falls in the center of a revolving cast-iron table and is moved toward the edge and finally scraped off by fixed plows, being moistened during its passage across the table by a spray of water. The result is very satisfactory, since there is no dust nor smoke and the product is in just the right condition as to moisture, being neither too wet nor too dry.

The Dwight-Lloyd product falls into side-dump railroad cars and is taken to smelting beds. Local conditions necessitate frequent changes of charge, so that the sinter is spread on smelting beds instead of being smelted separately.

This plant was one of the first to adopt the Dwight-Lloyd sintering machine, and while the management went through the usual experience that falls to the lot of beginners with a new process, it also had the pleasure of seeing the blast-furnace capacity increase, in the case of the 48 by 180 in. furnaces, from 180 to 250 tons per day for long periods.

Ore as received is handled as at all custom lead smelters, the only variation being that rather more screening is done in the case of oxidized ores, so as to send the coarse material to the blast furnace and the fine to the sintering machines.

While steam is kept up in one boiler for heating purposes and for some small machines, most of the power is purchased from the local

electric light and power company, which has a hydro-generating plant on the Little Arkansas River, a few miles above the town.

Water for boiler and jacket use is obtained from the Arkansas River, when it is clear; otherwise, from springs. The river water is better, carrying only 8.77 grains of solids per U. S. gallon against as much as 32.47 grains per gallon in some spring water. These springs are seepage springs caused by irrigating water leaching through the surface soil on the upper *mesa*. When spring water is used, the jackets must be washed once in 30 days.

The bare description of the plant with the dimensions of apparatus and appliances cannot be very instructive to beginners nor interesting for those who have been long in the work. However, it seems that some of the peculiarities of the Salida smelter as originally built should serve to prevent the younger men from making similar mistakes; the older men can get their satisfaction by reflecting that they never did anything quite so far removed from good practice.

Originally, a brick wall, with a narrow door in front of each blast furnace, was constructed. How the work was to be done behind this wall is not known, but it shows that the designer did not know how to do the work.

The water jackets were held in place by a 12-in. I-beam running the length of the furnace, thus effectually preventing any one from examining a tuyère or jacket.

The shaft walls on the feed floor were so thick that the feeders were shoving ore into one end of a tunnel, and, since the opening was not as wide as the shaft, the ends of the furnace could not be fed except by throwing the charge around a corner—which, of course, was not done.

The proposal to concentrate matte, carrying 20 per cent. lead and 5 per cent. copper, in a copper-matting furnace was intended to do away with the necessity of roasting equipment.

The largest Corliss engine on the plant was the exhibition engine at one of our world's fairs held some years ago. When repairs were needed, it was discovered that this was the only engine of the kind ever built, and the drawings of it were lost; so each repair part called for an accurate drawing.

In the track plan of the works, all the ore, coke, and limestone had to be unloaded from one track. This track also held the cars which were to be unloaded at the oxide crusher as well as at the sulphide mill. The problem of dispersing some of this congestion was not simple. It was solved by building two trestles in place of one; by having the limestone brought in dump cars so that unloading took little time; and by building an oxide crushing and screening plant on new tracks.

Originally the bullion was sampled by taking a dip sample from the lead well, which was not satisfactory. The present management built

drossing kettles in which the bullion is stirred by compressed air, skimmed into a Howard press, stirred by hand and siphoned out into bars for shipment, after sampling by dipping "gum-drops," approximating $\frac{1}{2}$ assay ton in weight. For some time the results were unsatisfactory. Although the copper had been reduced to 0.06 per cent., the bullion had a dirty appearance, and "gum-drops" of the same dip would not check with each other, especially in gold content. Consultation with men versed in the subject did not help. A remedy was finally found when the temperatures were determined by a pyrometer and the difficulties overcome by the following procedure: Take off heavy dross at 900° F.; blow until temperature reaches 680° F., skimming from time to time; raise the heat to 720° F. for sampling, and mold at 800° F. Since this method was adopted, little trouble has been encountered.

A plan to eliminate the personal equation in bullion assaying has been carried out for some years, with good results. The "gum-drops," weighing about $\frac{1}{2}$ assay ton each, are weighed without trimming, cupelled, and parted; the weights of the "gum-drops" in decimals of an assay ton, of the gold and silver together and of the gold alone, in milligrams, are reported by the assayer. The calculations are then made in the office on a machine. In this way the assayer is not tempted to make his results check, because until they are figured he does not know whether they will check or not. The excellent results obtained are sufficient reason for following the plan.

Referring again to the Dwight-Lloyd sintering plant, it must be borne in mind that this plant was built when there were no plants to serve as guides. The original charge-measuring and mixing apparatus is still used, although many accessories have been evolved since the construction of the plant. The charge is made up from six bins with belt bottoms discharging on to a belt that delivers the charge to the center of a cast-iron revolving table. The charge is moved to the edge of the table by fixed plows, water being added during the time of plowing; then it is scraped off into ore cars, sent to the top of the building on a platform elevator, and dumped into the machine feed bins. The use of a platform elevator instead of a bucket-and-belt elevator was decided on after seeing the segregation that took place when a belt elevator or conveyor was used. This stopped the segregation of coarse charge along the edge of the pallets.

For a long time coarse charge was fed to the pallets by means of an extra hopper behind the regular feed hopper, but it was found that if the back of the regular hopper was removed and the feed regulated to a thin stream, all the coarser particles would roll to the grate and stay there, stopping grate leakage and leaving only fines on top, presenting a proper surface for ignition. Coal ignition is still used because gas is not available and with a wooden structure gasoline is feared.

An important improvement has been made in the construction of the furnaces, by placing an adjustable rail by the side of the pallets so that the weight of the loaded pallets is carried on the rollers and not dragged across the top of the wind box causing excessive wear. The height of the rails is adjusted so that as the top of the wind box wears, the rails take the loads. This arrangement leaves no appreciable space between the bottom of the pallets and the top of the wind box.

Many small improvements are planned that will effect savings without great outlay of money. Like all custom lead smelting works, the Salida plant at times is called on to handle more zinc than is comfortable. The chief lead supply is that from the Cœur d'Alêne district of Idaho. Ores with iron in excess are obtained from Leadville, and the silica supply is helped out by receipts of clinkered zinc residue from plants of an allied company operating in Oklahoma.

Lead Smelting at El Paso

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THE lead department of the El Paso Smelting Works at El Paso, Tex., is operated strictly on a custom basis. The ores treated are drawn from the surrounding territory, comprising New Mexico, Arizona, western Texas, and northern Mexico. Ores received from the last-named section originally amounted to a very large tonnage, but of recent years have dwindled into comparative insignificance so far as this plant is concerned. This has been due partly to the establishment of a lead-smelting plant at Chihuahua, which is much closer than El Paso to most of the important mines, and partly to the recent revolutionary troubles in Mexico. There are no mines of either lead or copper in the immediate vicinity of El Paso.

In general, it may be said that the lead-ore supply of recent years has been uncertain and precarious, with a general tendency toward an excess of silica and a shortage of iron. This latter requirement has been met in large part by the use in El Paso lead furnaces of considerable amounts of roasted leady copper matte from the Chihuahua smeltery and of converter slag from the El Paso converters. These two materials, coupled with complex lead-copper ores from Bisbee, Ariz., have introduced into the lead-smelting operations a most unusual amount of copper, which has had an important bearing upon the smelting practice of the plant. This matter will be discussed later.

The ore, loaded in either box cars or gondolas, enters the storage yard on four tracks elevated about 8 ft. above the ground. As all unloading is done by hand, unloading platforms at the height of the car floor parallel the tracks on each side to afford a runway for wheelbarrows. At frequent intervals these lengthwise platforms are joined together by others running crosswise to facilitate the distribution of the ores evenly over the beds, which are located below and between the tracks. At present much of the bedding space formerly used for lead ores is given over to the copper department, but 20 bins of various sizes are retained for the use of the lead furnaces and the Huntington & Heberlein roasting plant. The largest of these holds about 1,000 tons, while the smallest holds not over

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175 tons, their combined storage capacity being about 10,000 tons. The bins have vertical wooden walls 8 ft. high and have no facilities for the removal of the ore by any mechanical means. Ore which for any reason cannot be accommodated in the bins is unloaded in stock piles in the outer yards and eventually loaded into railroad cars again with a Browning locomotive crane, which has been a part of the plant equipment for about two years past.

All unloading is done by contract, the rates per ton being based upon the character of the car from which the ore is taken, and the distance to which it must be wheeled. When, as is generally the case, the initial sampling of the ore is done by hand, the contract rate includes the cutting of the sample and the wheeling of it to the sample storage platform. An extra allowance is made for spreading the ore in the bin itself, which in most cases is carefully done by hand because of the widely different character of the various small lots which are bedded together. This last-mentioned condition is typical of the El Paso plant and the large number of small shipments has always been a serious obstacle in the way of installing a modern ore-bedding system.

In most cases it is customary to assign to each car to be unloaded a gang of four men, who cut with their shovels a one-tenth sample from the fines and set aside all lumps over about 6 in. for sampling in the automatic sampler, which occupies a space on the platform adjacent to the ore bins. This automatic plant is equipped with Vezin samplers, which are arranged to cut a one-tenth sample that may be combined with the shovel sample of the fines. The full equipment includes one 75-hp. motor; one No. 6 Gates crusher, capacity about 45 tons per hour; one 14-in. bucket elevator; one Vezin sampler, 52 in. in diameter over the wings, making a one-fifth cut; one set of 14 by 36 in. rolls, adjustable to $1\frac{1}{2}$ in.; one 10-in. bucket elevator; two smaller Vezin samplers, of which the combined sample cut from the stream is one-half.

The sample of the lumps from the automatic is added to the hand-cut sample of the fines and this combined sample is wheeled to one of the two smaller automatic samplers facing the unloading platforms and just inside the sample room proper. These two mills are practically identical. Their equipment includes one 9 by 12 in. Blake crusher; one 8-in. bucket elevator; one Vezin sampler cutting one-fifth; one set of 12 by 36 in. rolls; one 6-in. bucket elevator; and two Vezin samplers so placed that each one makes a one-fourth cut at the same point. These mills deliver two identical samples, crushed to $\frac{1}{2}$ in., which are generally finished up and assayed separately. These samples weigh from 400 to 500 lb., in the case of a one-car lot. After going through the finishing rolls, of which there are two sets, 14 by 24 in., the samples are wheeled to a separate compartment which has an iron-plated floor. There the samples are circled, coned, spread, and quartered by hand until reduced to a proper size to

send to the bucking room. A portion of the reject from the plates is set aside and held pending the final settlement with the shipper, in case any doubt may be cast upon the accuracy of the work done in the bucking room.

The foregoing is the usual method of sampling lead ores, although exceptions occur in the case of shipments of matte or coarse lump ore, and in the case of high-grade ores and concentrates. In the case of the former, the shovel cut is dispensed with and the whole carload is put through the large automatic sampler, while in the case of concentrates the procedure is reversed and the entire sample is cut by shovel one-tenth or one-fifth according to the grade, the sample being reduced entirely by coning, circling, and quartering. Very high-grade ores are usually cut one-fifth.

After sampling, all sulphide ores and leady copper mattes destined for treatment in the Huntington & Heberlein roasting plant are wheeled to the sulphide mill, which occupies a space on the unloading platform between the sample room and the roasters. This mill contains the following: One 12 by 18 in. Blake crusher; one set of 16 by 30 in. rolls, crushing to $\frac{1}{2}$ in.; one 12-in. bucket elevator; one $3\frac{1}{2}$ by 8 ft. trommel having openings $\frac{3}{4}$ by $\frac{3}{16}$ in.; one set of 6 by 42 in. high-speed rolls, to take the oversize from the trommel; one 8-in. bucket elevator, to return same to trommel again. This mill has a capacity of about 11 tons per hour.

The leady copper matte is generally bedded by itself, while the crushed ore is often bedded with sulphide concentrates, of which a varying tonnage is received at the plant. Because of the uncertain nature of the ore supply, it is difficult to say just what might be considered a typical mixture for the roasters. In general, the best results are obtained from a mixture of ore and leady copper matte, as straight ore is difficult to handle alone, and it has been found necessary to use at least 20 per cent. of siliceous sulphides to produce satisfactory results when roasting leady copper matte.

The ore and leady copper matte are trammed in 1-ton cars from the bins to the roaster plant, where the cars are elevated by a small hydraulic elevator to the level of the charge floor of the Godfrey roaster. The material is dumped into conical hoppers that feed mechanically into three Godfrey roasting furnaces, which are equipped with revolving hearths and stationary rabblers which plow the material gradually from the feed at the center to the discharge hole at the outer edge. All three furnaces were originally fired with coal which has since been replaced in the case of two of them by oil. The hot coals from the fire-box of the third furnace are used to start the Huntington & Heberlein pots. These Godfrey furnaces are 26 ft. in diameter and have a capacity of about 30 tons each per 24 hr. The oil-fired furnaces use about 0.22 bbl. per ton roasted, while the coal-fired furnace uses about 0.11 ton of coal, all of which, how-

ever, is not chargeable against direct roasting, as much half-burned material is drawn off for the pots. The initial charge probably averages 21 per cent. sulphur and this is brought down to from 10 to 12 per cent. in the Godfrey roasters. Care is taken not to make too strong a roast at this point, or the mixture will not work well in the pots.

The roasted material is slightly moistened by a spray as it leaves the discharge hole of the roaster and falls into a small hopper. From the hopper it is drawn into cars, trammed to the elevator, and raised to the charging floor of the Huntington & Heberlein pots. There are eight of these, each of which handles two charges of about 6 to 7 tons in 24 hr. At times it has been found possible to add to the pre-roasted material in the pots sufficient raw sulphides to bring the total capacity of the plant up to 100 tons per 24 hr. It is customary to work the pots in sets of four, so that when one set is charging the other is nearing the end of the blow. When the pots are filled, which takes about 2 hr. for the four pots, the full pressure of the blast is thrown on while at the same time the blast is cut very low on the four other pots which are finishing. Each charge is blown for about 11 hr., then dumped, and 1 hr. more is consumed in preparing for the next charge.

Air is furnished to the pots at a maximum pressure of about 10 oz. by a No. 7½ Baker blower, belt connected to a 35-hp. motor. The pots are suspended at a height of 10 ft. above the ground level and are dumped by a worm gear operated by hand. The sintered material is broken more or less by the fall, after which the breaking is completed by hand. The lumps are reduced to about 8 in. size, loaded into wheelbarrows, elevated to the level of the ore-unloading platforms by means of a small hydraulic elevator, and dumped into one of the regular ore bins convenient to the blast-furnace charge floor. The fines, which amount to from 5 to 10 per cent., are gathered up and returned to the pots on the next charge. The sulphur in the final product going to the bins averages about 5 per cent. The work of breaking and wheeling to the bins is done by contract.

The ore beds are grouped in the rear of the blast-furnace building and are connected with the weighing floor by a system of hallways, through which the ore is brought in wheelbarrows to a hopper standing over the larry-car tunnel. These tunnels, of which there are two, run at right angles to the line of blast furnaces and lead on to hydraulic elevators that raise the cars to the level of the feed floor, a lift of 41 ft. Because of the angle of approach it is necessary for the cars to run around a sharp curve after reaching the feed floor in order that they may pass lengthwise over all the furnaces in turn. As the elevators are approximately in the center of the building this operation involves a switch to reach the furnaces in the middle of the line. The larry cars are of the dump-bottom type, 5 by 10 ft., with a capacity of about 200 cu. ft., and are cap-

able of handling a 5-ton charge, although the charge actually in use at the present time averages about 4 tons dry weight including the weight of the coke. The cars are operated by 4-hp. motors connected with a third rail, and are hand dumped by a system of levers and counterweights.

The charge, which usually contains an average of about 8 per cent. moisture and is figured for about 7,000 lb. dry weight, is handled by wheelbarrow in units of from 400 to 500 lb., the latter being considered about the maximum load for the average wheeler. The various ingredients of the charge, after being carefully weighed upon ordinary multiple-beam platform scales, are distributed as evenly as possible in the charge hopper. This hopper receives the ore, both raw and roasted, and the limestone, which latter arrives at the plant in 50-ton dump-bottom railroad cars and is dumped through a trestle near the scales. After receiving the main portion of the charge from the hopper, the car moves under an open hatch about 15 ft. distant, through which are dumped the coke, scrap iron, dry dross, and miscellaneous furnace products that are being added to the charge. The coke is usually trammed direct from the cars in which it is received at the plant.

On the charge floor each furnace is provided with two parallel openings 1 ft. 4 in. by 11 ft., closed by counterweighted steel doors that may be easily opened by means of levers at the end. The charge car stops directly over the furnace and dumps its load, which is distributed by the spreaders below.

There were originally eight lead blast furnaces, but two of them were dismantled in 1913 to make room for an additional copper furnace.

The furnaces have the usual brick base 7 ft. 2 in. by 15 ft. 5 in. inside the crucible plates, which are 3 ft. 4½ in. high and stand upon a flanged steel pan. The crucible proper is 24 in. wide at the bottom, widening to 3 ft. 6 in. at the top. It is 12 ft. 9 in. long and 2 ft. 6 in. deep. The lead well is 12 in. square, and is built up 9 in. above the top of the crucible plates. It is on the right-hand side, looking toward the furnace, and its center comes 4 ft. 3 in. from the end crucible plate.

Resting upon the brickwork surrounding the crucible is the tier of lower jackets, which are 46 in. in vertical height and have a bosh of 9 in. at both ends and sides. The lower side jackets are of steel, all others, both upper and lower, being cast iron. Between the lower and upper tier is a 3-in. layer of firebrick. The upper jackets are 24 in. in height without bosh. Each of the lower side jackets, four on a side, has two 3½-in. tuyères placed 13 in. from the center of the hole to the bottom of the jacket. The hearth area of the furnace at the tuyères is 46 by 162 in., or 47¼ sq. ft.

Supported by brick arches which rest upon 8-in. iron columns at each corner of the furnace is the brick shaft, 10 ft. 8 in. by 20 ft. 4 in., outside

measurement, and extending upward a height of 23 ft. to the feed floor. Inside the shaft the furnace widens from 5 ft. 6 in. at the top of the jackets to 7 ft. 1 in. at a point 6 ft. below the charge floor, which is about the level at which the ore column is usually maintained.

The furnaces average about 180 tons each of total charge per furnace day, although with a coarse charge they have frequently done better than 200 tons for considerable periods. During recent years not more than three furnaces have been in operation at any one time.

At the rear of the furnace just under the floor plates, a rectangular steel downtake, 3 by 6 ft., conducts the gases to the brick flue chamber, a vertical distance of 11 ft. 4 in. and a horizontal distance of 12 ft. 4 in. The flue system, which varies somewhat in the different sections, has an area of 151 sq. ft. at its smallest cross-section and 170 sq. ft. at the largest. Its total length is about 800 ft., of which the first 375 ft. nearest the furnaces is provided with rectangular iron hoppers overhead that discharge into tram cars on a track within the walls of the flue chamber. The rest of the flue is provided only with iron doors at intervals along the sides at the ground level, through which the flue dust must be raked out. By far the greater amount of dust is deposited in the hoppers, from which it is removed daily, while the rest of the flue is ordinarily cleaned only once in three months.

The production of flue dust is not large, averaging from $1\frac{1}{2}$ to 2 per cent. of the charge smelted. The following average analysis for a recent month gives a fair idea of the ordinary composition: SiO_2 , 16.8; Fe, 17.6; Mn, 0.6; CaO, 8.1; Zn, 4.0; S, 9.0; Pb, 18.2; Cu, 1.4; Au, 0.30; Ag, 23.5. The flue dust is briquetted with 10 per cent. burned lime in a Chisholm, Boyd & White press, which has a capacity of about 4,000 briquets or 6.5 tons per hour. The briquets are stacked on wooden pallets, 300 to the pallet, and are taken to the drying shed on Scott briquet cars, which are so arranged that the carrying platform may be raised and lowered by a lever so as to permit the car to be easily withdrawn from under the pallet, leaving the latter resting on the stands. The shed has a storage capacity for 425 tons, which is sufficiently large to permit the briquets to be left to dry for from three to six weeks. After this length of time they are exceedingly hard and reach the furnace in very good shape.

For a long time past the settling and separation of the leady copper matte from the slag has been a most vexing question for the El Paso plant. This was originally accomplished by the use of a Rhodes "separator," or small reverberatory settling furnace, which in the days of large tonnage was generally satisfactory. However, with the present comparatively small tonnage of lead ores smelted it has proved increasingly difficult and expensive to successfully operate these furnaces and they are now permanently abandoned. In spite of this, the "separators"

were for so long a distinctive feature of the plant that it may not be out of place to give a brief description of them and of the details of their operation.

There were two of these furnaces placed end to end in a line parallel with the line of blast furnaces and about 100 ft. distant from it. It was customary to operate only one separator at a time, the other being held in reserve. The slag and leady copper matte were tapped from the lead furnaces directly into small pots, which were then pushed by hand along iron-plated runways to the dump hole of the separator. Also for many years all the slag from the settlers at the copper blast furnaces was put into the separator, partly because it helped to keep the furnace in better condition and partly because there was no other easy way of disposing of it.

These settling furnaces had an effective inside area 14 by 25 ft., with rounded corners at the end opposite the firebox. The whole structure rested upon a steel pan supported by 6-in. I-beams. Above the pan came 9 in. of tamping, in which was imbedded a continuous coil of 1½-in. pipe covering the entire surface. Above the tamping was 9 in. of firebrick, laid in the form of an inverted arch with a dip of about 2 in. in the middle. The walls were 1 ft. 8 in. thick and contained a tier of cast-iron coil jackets, with a lining of 4½ in. of firebrick inside the jackets. The roof was formed of a 9-in. firebrick arch and sloped downward from the bridge wall toward the uptake to the stack at the opposite end. The dump hole was at the corner close to the uptake; the slag tap on the same side as the dump hole, but at the opposite end of the furnace near the bridge wall; and the matte tap just about opposite the slag tap on the other side of the furnace. A lead tap was also provided close to the matte tap on the same side. The ordinary depth of the bath of molten slag and matte was about 36 in. The center of the slag tap hole was 28 in. above the brick bottom, the matte hole 15 in., while the lead tap was level with the bottom at its deepest part. Both slag and matte taps were water jacketed. The slag was tapped direct into oval-bowl dump cars of about 3 tons capacity and hauled to the dump by a small steam locomotive. The leady copper matte was tapped into a rectangular cast-iron box 3 ft. 8 in. by 6 ft. 2 in., which served as a trap to catch any metallic lead that might come with the matte. From the box two cast-iron spouts conducted the matte to the edge of the granulating pit, which was a cement-lined reservoir constantly filled with water and from which the granulated matte was pulled with hoes. The two streams of matte falling from the spouts were cut by a double spray of water at about 100 lb. pressure, two nozzles being provided for each stream as a safety measure in case one might suddenly become clogged up. The granulated matte was wheeled to railroad cars and transferred to the Huntington & Heberlein roaster

plant, where it seemed to give quite as good results as matte that had been crushed in the sulphide mill.

The furnaces were originally coal fired, but during November, 1911, they were changed over to oil. At first the burners were introduced through the firebox and played over the bridge wall, being inclined slightly downward and exercising a reverberatory action. Afterward this arrangement was changed and the burners set to play directly downward through the roof, one at each corner of the furnace. This was a decided improvement and kept the furnace in much better condition with less oil. The amount of oil used varied considerably, but 20 bbl. per day was about the average.

The advantages of the separator were the almost perfect settling of the leady copper matte under normal conditions and its cheap preparation for roasting by means of granulation. The disadvantages were much more numerous. To begin with, the transportation of the material from the blast furnaces in hand pots was expensive. The practice of mixing lead and copper slags together in the presence of leady copper matte was eventually found to be a serious mistake and steps were taken to dispose of the copper slag elsewhere. It was conclusively proved that the copper slag saturated itself with lead and silver and thus carried to the dump valuable metals that would otherwise have been saved. The effect of cutting out the copper slag, the volume of which at times was considerable, was to leave the separator an easier prey to the troubles brought on by speiss crusts and zinc mush, and to increase the amount of fuel necessary to keep the bath in good condition. It was found that high zinc in the charge would fill up the separator with "mush" quite as quickly as it would a smaller settler and that a few days of exceptionally strong reduction on the blast furnaces would often produce layers of speiss that would set and form pockets and false bottoms that were equally fatal to the settling action of the furnace. Another peculiarity was that the leady copper matte invariably came out with a higher lead assay than it had when it went in. This undesirable condition was thought to be due to a reaction between comparatively large surfaces of molten matte and metallic lead, of which there was always more or less at the bottom of the furnace.

It was never found practicable to operate the separator with less than four men on a shift, and when the lead plant got down to a two-furnace basis the amount of fuel burned was much greater than it would have been had a larger volume of molten material been handled through the settling furnace. The length of the campaigns of the separators between shutdowns for extensive repairs became shorter and shorter until in July, 1914, a newly repaired furnace became a hopeless mass of crusts and pockets within a week of the time it was put in use. The other furnace was in similar shape and the plant was left without any settling

device except small rectangular cast-iron boxes 3 ft. 4 in. by 6 ft. 4 in. and 2 ft. deep, which were entirely inadequate even when run in tandem. At this time every one connected with the plant became convinced that the separators had outlived their usefulness and that a change was imperative, but the whole blast-furnace system was linked to these furnaces in such a way that a change to any standard plan could be made only at an excessive cost and an unjustifiable interference with the operations of the plant in general. One serious difficulty was the lack of sufficient elevation in front of the blast furnaces, the space between them and the separators being comparatively level and only 4 ft. below the furnace tap holes. This was practically all made ground, having been built up of loose boulder slag, which made excavation difficult and dangerous on account of its tendency to ravel and cave in. Another trouble was the lack of room for any arrangement of tracks such as are generally to be found in front of a line of modern blast furnaces.

It was decided, therefore, to proceed along somewhat original lines, and on July 25, 1914, there was put into operation in front of one furnace a round, oil-fired settler, 8 ft. in diameter. This was intended to serve merely as a second settler to remove the last traces of matte from the slag before going to the dump. For the main separation one of the old rectangular boxes, already mentioned, was relied on, being placed between the round settler and the furnace. Results obtained from this settler were not particularly satisfactory as numerous particles of matte persisted in floating across the surface of the bath direct from intake to outlet. Also, the settler shortly began to fill up with a zinc mush that interfered seriously with proper settling. While the slag that was being run at the time contained only 4 per cent. of ZnO , there seemed to be present in the Huntington & Heberlein roast enough unaltered zinc sulphide to give trouble. The furnace to which this settler was attached was blown out soon afterward and was replaced by another furnace, in front of which was a similar arrangement of box and round tank, except that in this case the latter was 10 ft. in diameter. The tank was set off center far enough to give the furnace tapper a chance to work, which made necessary a rather long curved spout from the box to the tank. The greatest trouble encountered was the lack of elevation, as the round settler was erected on the original level of the ground in front of the furnace and it was necessary to have both slag and matte taps high enough for the hand pots to get under them. It was thus necessary for the slag to flow a total distance of about 23 ft. with a drop of only 11 in. between the tap hole and the outlet from the settler. This of course caused trouble every time the slag happened to get thick and pasty. The first settler, or box, was easily removable, had a firebrick lining (magnesite was afterward used with great success), and measured 66 in. long, 28 in. wide, and 21 in. deep inside the lining. The round settler had a 9-in. lining of

magnesite brick, both sides and bottom, and a 9-in. arched roof of firebrick, upon which rested at the edge of the tank, just over the slag outlet, a short brick stack with an iron pipe connection above. The oil burner was placed at the exact center of the roof and was directed straight down upon the surface of the bath. A small combustion chamber was later placed on the roof and proved very successful in getting more efficient results from the oil.

It was thought that a baffle extending across the tank close to the inlet and partly immersed in the bath might compel the floating particles of matte to go down and join the main body of matte instead of going straight across and out with the slag. Accordingly, such a baffle, constructed of brick, was placed about 12 in. from the inside of the lining. This feature of the experiment was a complete failure and the floating specks of matte continued to pass out on the surface of the slag as before. These small particles, however, proved upon investigation not to involve so serious a loss as had been feared. Several hand pots of slag were set aside to cool, each of which contained a plainly visible number of the specks. After cooling, the balls were broken up and put through the small automatic sampler. The average results were: Silver, 0.65; wet lead, 1.00; copper, 0.15 per cent.

After starting off favorably, this new settler soon began to fill up with zinc mush. Some of this material had a specific gravity lower than that of the slag, while some of it appeared to be slightly heavier, the tendency as a whole being to form a layer between the slag and the matte. Various samples of this material were analyzed, a typical one being as follows: Insoluble, 6.6; iron, 19; lime, 3.4; zinc, 24.3; sulphur, 21; lead, 10; copper, 4.6 per cent. When this crust was broken up with a bar, it floated for a while at or near the surface in lumps resembling dry dross. This was not a new condition, having frequently been observed in the Rhodes separators. The only way to overcome it seemed to be to let the settler fill up with matte, thus forcing the mush to the top, where it was skimmed off with rabbles through a hole cut for the purpose. This of course involved a considerable loss of matte and a large proportion of the slag made at this time had to be saved for resmelting. This condition of affairs was finally remedied by changing the charge so as to use less of the Huntington & Heberlein roast. After this time similar troubles were promptly met in their early stages by changes in the furnace and roaster charges and never again caused serious trouble while this style of settler was being used.

As already mentioned, the original idea with regard to this type of settler was to use it simply as a safeguard against letting any matte go to the dump, as much matte as possible being tapped directly from the box. It was found that this amounted to about 88 per cent. of the total matte produced. The shells from the hand pots in which the slag was

removed were frequently and carefully sampled, and they assayed so little higher than the regular slag samples that they were thrown away. The ordinary method of taking the regular daily samples was to take a small ladleful from the stream as it flowed from the round settler, once every hour, and to pour it into water for granulation. At the end of 24 hr. the whole sample was worked down for a pulp for the assay office.

A second furnace was blown in Sept. 29, 1914, with a settling arrangement exactly similar to the one then in use. The two furnaces were operated throughout the month of October in the manner already described. The average analysis of the slag for the month was: SiO_2 , 33.7; FeO , 27.9; MnO , 1.4; CaO , 22.9; ZnO , 4.6; Al_2O_3 , 5.2; Ag , 0.6; wet lead, 11.50; Cu , 0.15 per cent. The leady copper matte produced averaged Fe , 42.1; Pb , 12.1; Cu , 19.8; Zn , 4.6; S , 21.4 per cent.; Au , 0.04 oz.; Ag , 87.8 oz. The average grade of the bullion produced was: Au , 3.24; Ag , 247.6. The matte produced averaged 8 per cent. of the total charge. Lead in the charge averaged 15.7 per cent., and sulphur 2.2 per cent. Coke used averaged 14.2 per cent. These figures, while applying to only a single month, may be taken as fairly representative of the lead-smelting practice of the plant.

The high copper in the leady copper matte, as shown in these figures for the month of October, is characteristic of the usual conditions at this plant. It had long been customary to set aside matte of about this grade until a sufficient amount had been accumulated to make worth while a concentration run. The matte produced during these campaigns usually averaged about 10 per cent. Pb and 35 to 40 per cent. Cu . It was drawn off in hand pots, cold dumped, broken by hand, and resmelted through the copper blast furnaces. All this involved considerable expense and it became evident that much of this could be saved if some means were provided for getting this molten leady copper matte direct to the converters. The round oil-fired settlers offered a convenient reservoir in which to collect the matte in fairly large quantities, and a pit 10 ft. wide and 8 ft. deep was dug immediately in front of the settlers in order to allow the approach of the 5-ton matte ladles serving the converters. About this same time the rolling stock equipment of the plant received a much needed addition in the shape of several Treadwell slag cars of 160 cu. ft. capacity, holding about 14 tons of slag each. These cars superseded the older ones of small capacity and made it possible to handle all the slag and matte away from the furnaces over a single straight track laid in the pit in front of the furnaces, entirely doing away with the necessity for hand pots for any purpose. This, however, involved a radical change in the use of the settling tanks, inasmuch as all the leady copper matte was run over into them and the rectangular box was used only as a collector for lead. Owing to the high percentage of copper on the charge and the large amount of dross produced, the lead wells of the furnaces are very

hard to keep open, and most of the time all the lead passes out of the tap hole and must be separated outside the furnace.

The direct conversion of the leady copper matte proved to be such an economy that since the introduction of this system, matte has been converted that is much lower in copper than formerly. At the time of writing it is customary to figure the value of the lead contained in the leady copper matte, and therefore lost in converting, as against the cost of roasting and resmelting in the lead furnace, taking into consideration the fact that each unit of copper present means the loss of a certain amount of lead even on a concentration run. Each day's production of leady copper matte is thus disposed of according to its assay, either direct to the converters or poured on the ground in beds, to be picked up, crushed, and roasted. As the matte has recently been rather low in lead, most of it has gone direct to the converters even though it may not run higher than 20 per cent. copper.

Under this system the whole duty of settling out the matte is put up to the large settler, and the rectangular box is used only to separate out the lead. Considerable anxiety has been felt as to whether it is safe to rely upon a single settler for the separation of the matte. In an endeavor to throw some light upon this question, the shells from the Treadwell pots have frequently been brought back from the dump and sampled with great care. The most complete of these tests covered a period of five full days early in January, 1915, all the shells produced being saved and run through the automatic sampler. The resulting composite sample showed: Au, trace; Ag, 0.64; wet lead, 1.46; Cu, 0.30 per cent. The average of the daily 24-hr. samples for the same period showed: Au, trace; Ag, 0.30; wet lead, 1.24; Cu, 0.24 per cent. These figures, which are about in line with all the others obtained under similar conditions, would indicate that it is safe to put these shells over the dump.

The comparative success attending the use of these round oil-fired settlers encouraged the installation of a somewhat similar but larger settler intended to serve two furnaces instead of one. The apparent advantages of this were the greater volume of leady copper matte that could be accumulated at one time, the certainty that one such settler would require less fuel oil than two of the earlier type, and some saving in labor.

This new settler is placed about 6 ft. forward of the line of the front crucible plates of the two furnaces and equidistant from them. It is 10 ft. wide by 20 ft. long, extending close enough to the edge of the train pit to make it easy to tap matte and slag into the cars below without the use of excessively long spouts. The whole structure is supported upon rails running crosswise and providing air spaces beneath the bottom pan. This feature has proved to be of considerable importance inasmuch as there is a constant seepage of lead through the bottom and the open air spaces arrest its progress downward into the foundation and facilitate its

removal. The ends of the settler are slightly curved in order to give a stronger structure. The sides are braced with I-beams set upright and tied across the top with steel rods. The sides and ends have a lining of 9 in. of magnesite brick. A matte tap is provided on each side close to the front, and there is a single lead tap in the middle at the rear end. This lead tap is used every day, but very little lead is obtained, as experience with the Rhodes separators indicated the inadvisability of maintaining a bath of lead in contact with the matte, and every effort is made to keep lead from getting into the large settler in the first place. The roof of the settler is sectional, being composed of a number of rail-bound firebrick arches that can easily be lifted off if necessary. The stack is situated at the rear end immediately between the slag inlets from the two furnaces and over the lead tap. Two oil burners were installed, one playing downward through the roof at a point about 6 ft. back from the front, and one playing nearly horizontal through the front wall.

Between each furnace and the large settler is a removable rectangular box, 2 ft. 6 in. wide by 9 ft. 2 in. long, lined with $4\frac{1}{2}$ in. of magnesite brick all around. The furnace is connected with the box and the box with the settler by short water-coil spouts. Each box is set at an angle of about 45° to the furnace, giving the tapper a clear field for his work.

As it is no longer necessary to provide for the use of hand pots under the slag and matte spouts, it has been possible to arrange the elevations of the various parts of the system to much better advantage than in the case of the round settlers. There is a difference of 20 in. between the tap hole of the furnace and the bottom of the slag outlet from the main settler, which greatly facilitates the handling of the spouts. There is a difference of 10 in. between the bottom of the inlet and the bottom of the slag outlet, which allows plenty of time for switching and changing slag cars after the outlet hole has been closed up. The normal depth of the bath in the settler is 25 in. with the slag outlet open, and it is customary to maintain about 10 to 12 in. of matte at all times, as this seems to assist materially in keeping the settler in good condition.

Although this settler was designed for operation with two furnaces, it was decided to try it out with only one, in order to see if it might be kept open under such circumstances. At the time of writing it has been in successful operation with one furnace for exactly one month, although the amount of fuel oil necessary to keep it in good condition is somewhat greater than was the case with the former round settlers. The larger settler requires about 7 bbl. of oil per day, whereas the others averaged about 4.5 to 5 bbl. However, with two furnaces running it is probable that less oil would be required. No serious troubles have been encountered up to the present time, and careful sampling both at the furnace and on the dump indicates that the settler is standing up to its work quite well.

At such times as the lead wells are in working order, the lead is drawn

off into cast-iron pots at the side of the furnace, drossed by hand, and the dross returned directly to the furnace. When, as is more frequently the case at present, the lead wells are not working and all the lead is taken from the box, no attempt is made to dross it at the furnace. Instead, the undrossed bars are trammed to a small sweater furnace, where they are remelted upon a sloping hearth, the clean lead drained off, and the dry dross raked out and returned to the furnace charge. This equipment is inconveniently situated and antiquated in the extreme, and it is hoped will be speedily abandoned.

Whether drossed at the blast furnace or at the remelting furnace, the bullion is sampled in the same manner, as follows: The bullion is tapped into a cast-iron cooling pot. The dross is skimmed off carefully and thoroughly. Before the final skimming the contents of the kettle are well stirred with the skimmer and the dross thereby raised is taken off completely. After skimming, the contents of the cooler are stirred again and, while the lead is still in motion, "gum-drop" samples are dipped. A few more gum drops are taken than it is expected to make bars of lead, to insure having a gum drop for each bar of lead. The bullion is ladled into molds, and after the pot is empty all extra gum drops over the number of bars produced are thrown out. These gum drops are held and delivered to the assay office whenever the bullion corresponding to them is removed.

A shipping lot consists of 780 bars (about 42 tons), and, when complete, the 780 gum drops corresponding are placed in a graphite crucible and melted at as low a temperature as will give a homogeneous mass from which to dip the final gum-drop samples, upon which the actual assay of the lot is made. Great care is taken not to get the sample hot enough to oxidize the lead during the process.

The gum drops, which weigh about 40 g. each, are assayed without clipping and the gold and silver calculated to an assay-ton basis.

The Advantages of High-Lime Slags in the Smelting of Lead Ores

BY S. E. BRETHERTON,* SAN FRANCISCO, CAL.

(San Francisco Meeting, September, 1915)

DURING the year 1878-79, Anton Eilers, who was then interested in the lead smelting and refining business near Salt Lake City, Utah, made a somewhat radical departure from the regular practice at that time, which was to use but little lime in the slag, with a high percentage of iron. Lime was not only a cheaper flux than iron, but it enabled the metallurgist to make a more siliceous slag, an economical advantage in smelting where the smelting companies had to purchase either iron or lime to neutralize the excess silica, and penalize the ore shippers accordingly. The larger amount of both lime and silica in the slag also made it of lighter specific gravity, and a better separation of the metals and mattes was secured. This change in the formation of slags for lead smelting was brought about by Anton Eilers, prior to the Leadville mining excitement of 1879. We must also give Mr. Eilers credit for the general introduction of the hollow water-cooled cast-iron jacket for the blast furnace.

Again referring to the use of more lime and less iron in the formation of slags, there is no doubt but that it prevents the formation of a great many so-called "sows" in the lead crucible, such as were made in the early history of lead smelting in the Rocky Mountains; these sows consisted of lead, iron, arsenic, sulphur, copper, and other metals, including gold and silver, mixed with unconsumed fuel, and often, after vain attempts to cut them up or melt them to a suitable size for handling, were buried in the slag dumps to get them out of sight.

Another advantage derived from the adoption of high-lime slags when smelting ore containing arsenic is the production of less speiss (a by-product heavier than matte, resulting from careless operation of the furnace, in the lead being driven out of the crucible and sows built up). This speiss, while usually not as rich in precious metals as matte made at the same time, is too valuable to be thrown away; but it is difficult to treat, and it was thrown away or piled on the slag dumps in the pioneer days of lead smelting in this country.

About the time of the introduction of the lower iron and higher lime slags in lead smelting, sulphide ores were encountered in Utah and Colo-

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rado. Matte was made as a byproduct in smelting and arsenic gave less trouble. The general supposition was that the arsenic was taken up by the matte and finally lost in the treatment of the matte. No doubt this was true to a great extent and I gave the matte the credit of taking care of the arsenic until I had to do some special lead smelting in Nevada. The ore to be smelted contained an average of about $3\frac{1}{2}$ per cent. lead, $6\frac{1}{2}$ per cent. iron, $2\frac{1}{2}$ per cent. manganese, 7 per cent. zinc, 21 per cent. lime, 11.7 per cent. magnesia, $8\frac{1}{2}$ per cent. silica, 1 per cent. alumina, 3 per cent. arsenic, and only a trace of sulphur, together with a very small amount of ore, which we called our lead ore, from another mine averaging $11\frac{1}{2}$ per cent. lead, 20 per cent. iron, 4 per cent. zinc, 3 per cent. lime, no manganese, no magnesia, 30 per cent. silica, and only a trace of either arsenic or sulphur. From this it can be seen that there was more than enough arsenic to make a large proportion of speiss (especially as we smelted regularly considerable speiss containing gold and silver from some old slag dumps in the neighborhood), and no sulphur to make a matte to absorb it. At the same time, the bulk of the ore being so low in iron and manganese and high in lime and magnesia, I was compelled to make a slag very rich in lime and magnesia. The result was that we produced no speiss or matte, excepting once for a few hours when we fed in some sulphide ore to see what the results would be. During this smelting campaign of a little over one month, we had no trouble except with machinery and from shortage of fuel, which forced us to bank the furnace at one time for 24 hr. The result was \$36,000 worth of argentiferous lead bullion made in a short smelting campaign without a single "freeze up" or "blow out" and no speiss product. The only detail in our practice which differed from the ordinary method of smelting was to cupel our bullion for shipment until it was rich in gold and silver, so as to use the lead as litharge over again in the charge. The fact that we were able to get rid of the arsenic in the ore and speiss from old slag dumps and make a comparatively clean slag led me to believe that the high percentage of lime base in the charge and the lack of iron with which the arsenic could combine, resulted in the volatilization of the arsenic, this action no doubt being assisted by the oxidizing influence of the litharge used. While our gold and silver losses were satisfactorily low, our lead losses were very high, which was to be expected when using the lead over and over again in such an oxidized mixture and keeping the coke consumption as low as possible. Coke was the only reducing agent used.

While making matte in the blast furnace in New Mexico and Arizona from low-grade copper ore containing plenty of sulphur and sometimes as much as 4 per cent. arsenic in the charge, no arsenic was found in the matte, either by our chemist or the purchaser, as it had all been volatilized.

Returning to the subject of lead smelting, I do not know of any im-

portant improvement in the composition of slags since lime slags were introduced by Mr. Eilers.

As the oxidized lead ores of the Rocky Mountain region became exhausted and "sulphides" were mined from the deeper mines, both silver and lead losses became greater, as no provision had been made for the proper separation of the matte from the slag at that time. For several years the slag was tapped into the ordinary small slag pots. The losses were very high by reason of the fact that much of the matte was mechanically held in suspension in the slag, the small pots and the quick cooling of the slag preventing proper settling. Further loss was caused by carelessness on the part of the furnacemen, slag being dumped without having been cooled and broken up to save the matte. Then came the introduction of square cast-iron settling boxes on wheels, which are still in use, but much larger and better constructed than at first.

The production of matte containing a little copper, if allowed to settle from the slag properly, produces a slag lower in value; at the same time the matte fall, in addition to the lead fall, increases the smelting capacity of the blast furnace, and requires a lower percentage of fuel. The smelting capacity of the lead blast furnace was also increased, sometimes doubled, by increasing the depth of the ore mixture in the furnace and increasing the blast pressure. This increase in depth and blast pressure was not practical as long as the metallurgist held to the idea that part of his fuel must be charcoal, as this would crush and pack under such a heavy burden, the fine charcoal and high-pressure blast causing "over" or "top" fires in the furnaces, increasing losses by volatilization and burning the fuel before it reached the proper melting zone of the furnace.

Improvements have also been made in the more economical handling of the ore, permissible only when handling large amounts, such as belt conveyors from crushers to storage bins, automatic dumping cars for feeding the charges into the furnaces, etc.

As the amount of sulphide ore to be smelted in the lead furnace increased, the matte settlers were improved. The sulphide ore was also ground fine and roasted before smelting. The fine roasted ore was found objectionable, causing too much flue dust and packing of the charge in the blast furnace. Briquetting was tried, but was found unsatisfactory and expensive. Then what was known as the Omaha & Grant roasting furnace was designed to roast and then fuse the fine ore into a pasty mass in the same furnace. This made an ideal product for the blast furnace when broken up properly. This method was used for years, but the loss in lead was excessive. The next and one of the most important advances in lead smelting was the introduction of the Huntington-Heberlein process, or "pot roasting," which reduced the cost of roasting and the lead loss, at the same time producing a suitable product for smelting in the blast

furnace. Following this improvement in the metallurgy of lead came the Dwight-Lloyd sintering machine, a cheaper method of doing the same work.

I think I have mentioned the most important improvements in lead smelting for the production of bullion containing gold and silver. Lead as a collector of gold and silver from very rich oxidized ore has its advantages; the loss of gold and silver in the refining of the lead bullion is smaller than in the refining of copper matte, and when the ore contains sufficient lead it is the only process to adopt. Lead smelting requires not less than $12\frac{1}{2}$ per cent. lead in the charge, so that the concentration cannot be greater than eight of ore to one of bullion. If sulphides are to be smelted, the ore requires a comparatively sweet roast, and in the blast furnace three to five times the fuel is required that is necessary in the production of copper matte. In the blast furnace using ores containing copper sulphides as a collector less fuel is required, roasting is not necessary, more siliceous slags can be made, and a much higher concentration is possible by reason of the fact that a much lower percentage of copper is required as a collector. The preliminary removal of the zinc from a lead-zinc ore permits of an improvement in lead metallurgy. This subject has been quite fully covered in a previous paper¹ by me and in articles by F. L. Wilson and others.

The methods of sampling ores today are practically the same as those used more than 30 years ago. At large copper-smelting plants they have adopted automatic samplers to cut expense where the ore not only comes from their own mines, but is comparatively low grade. In custom plants where small lots of extremely rich ore are sampled and followed by samples taken from large lots of low-grade ore, automatic sampling machinery is not reliable. Recently I had an opportunity to observe the sampling of a large lot of rich ore at the Selby Smelting & Lead Co.'s works and I could not ask for a better system of sampling. I will not attempt to describe the method here except to say that it was not done automatically and was expensive, but absolutely fair to both the shipper and the company.

I have not touched on the subject of lead refining for the reason that in the West we are more interested in the production of lead bullion containing the precious metals. The bullion can be shipped to refiners in the East to better advantage than to refine it here and ship the metals separately.

In conclusion I wish to call attention to the following items in this article:

1. The advantage of high-lime and low-iron slags, especially when smelting lead ore containing arsenic.

¹ The Treatment of Complex Ores by the Ammonia-Carbon Dioxide Process, *Trans.*, xlix, 802 (1914).

2. The advantage of enriching the lead bullion by cupelling so as to use the lead over again as a collector of the precious metals; also the saving in freight on lead in isolated districts where the freight costs amount to nearly as much as the lead is worth.

The cupelling furnace referred to is such as I used in Leadville, Colo., and in Nevada, large enough to cupel 8 tons of bullion in 24 hr. It would also be suitable for the saving of antimony, thus shipping cleaner lead.

DISCUSSION

ANTON EILERS, Brooklyn, N. Y. (communication to the Secretary*).—While the advantages in the use of high-lime siliceous slags in lead smelting which Mr. Bretherton discusses in this paper are obvious under certain economical conditions, he gives me credit for the introduction of such slags when it does not belong to me. It is true I did considerable experimental work, off and on, for the purpose of finding such slags, while at the Germania in 1877 and 1878, but I did not have occasion to introduce any such slags for regular work, because I had no ores which fitted the economical conditions for using high silica and high lime, except in isolated cases for very brief periods. On the contrary, the ores I had to treat were mostly very ferruginous and very low in silica, so that, economically, I was compelled to run nearly always on slags, which allowed me to use the abundant flux I had without cost.

It was August Raht who first had occasion to establish and use for long periods a slag high in silica and lime, and low in iron. This was when he was confronted with the task of smelting the Horn-Silver ores at the smelting works at Franklin, Utah, with as little expenditure for fluxing material as possible. I was at that time in Leadville, Colo., where, in the new works I had built, the Billings and Eilers smelters, I had occasion to use profitably many slags, but never anything as high in the earthy bases and in silica as Mr. Raht had to use.

On Dec. 4, 1881, Mr. Raht sent me the following two analyses of slags, taken from the regular run on different days, and the remarks accompanying:

	I Per Cent.	II Per Cent.
SiO ₂	33.9	35.0
FeO.....	26.8	25.4
CaO.....	26.1	24.9
BaO.....	Not det.	3.2
Al ₂ O ₃	10.8	9.3
	—	—
	97.6	97.8
Pb.....	0.6	0.8
Ag.....	0.0	0.2

* Received Sept. 8, 1915.

- I. Produced with coke = 15.2 per cent.
 Bullion 163 oz.; lead contents of charge, 18.7 per cent.
 Blast pressure, $5\frac{1}{2}$ oz. per square inch.
- II. Produced with coke = 14 per cent.
 Bullion 152 oz.; lead contents of charge, 20.1 per cent.
 Blast pressure, $6\frac{3}{4}$ oz. per square inch.

This was a remarkably clean slag and, according to Mr. Raht, kept the furnaces in excellent condition and allowed of good tonnages. I looked up the nearest chemical formula that I could derive from the above percentages and found the following:



Assuming that the slag consisted only of silica, iron protoxide, and calcium oxide, it gives the following percentages:

SiO ₂	35.28
FeO.....	31.76
CaO.....	32.94
	<hr/>
	99.98

and assuming that 12 per cent. of iron and lime was replaced by foreign bases (in this case principally Al₂O₃ as given by Mr. Raht) the following percentages result:

SiO ₂	35.28
FeO.....	25.87
CaO.....	26.83
R.O.....	12.00
	<hr/>
	99.98

I therefore took this slag of Mr. Raht's to be a "correct slag" with all the virtues that correct slags have.

It will be noticed that it is a singulo-silicate of iron and lime plus a bisilicate of iron.

I occasionally used this slag, later in life, with advantage in both lead and copper smelting.

Mr. Bretherton also kindly credits me with the introduction of cast-iron water jackets. Cast-iron water jackets closed on top with an outlet pipe for the hot water on the side near the top (the inlet pipe for the cold water being near the bottom of the jacket) were in use at a few smelting works when I came to Salt Lake City, but frequent burning out on account of steam spaces forming, made their use rather costly. Together, Augustus Steitz, of St. Louis, and I devised a jacket with an open neck 7 in. high, of box shape, on the top of the jacket, into which the cold-water pipe reached down 10 or 12 in. and the hot-water outlet pipe

was very near the top of the neck, thus preventing any steam spaces in the jacket. It is true this jacket was generally introduced where cast-iron jackets were used. Some people, however, used costly wrought-iron jackets in preference, the reason for which has never been apparent to me.

The advantages which Mr. Bretherton sets forth, derived from the use of high SiO_2 and high CaO slags in general, coincide with my own observations. Of course, the main reason for using such slags must always be the economic side in the selection of flux, but the other incidental advantages which he mentions are also of great moment. The elimination of arsenic in certain charges can, according to my experience, be increased by cutting the fuel percentage slightly below that needed for perfect reduction, and by giving, at the same time, say, 1 per cent. more CaO than the formula requires. This, however, may increase the silver loss slightly.

Some Problems in Copper Leaching*

BY L. D. RICKETTS, NEW YORK, N. Y.

(San Francisco Meeting, September, 1915)

IN recent years the metallurgical field of the copper industry has expanded greatly, the copper ores have become lean and diverse in character, and we are obliged to treat such ores on a very large scale.

On the commercial side, the operating and consulting staffs of the great mining companies have been enlarged and organized, and no great capital expenditure is decided upon until it has been investigated and recommended by a highly organized force of specialists as a committee.

I believe that in the scientific field and in the most valuable proceedings of our various societies the old idea of one-man point of view and one-man treatment of principle and detail together has been splendidly developed and should continue, but for the advancement of our art we should deliberately discuss subjects among ourselves in order to do away with the idiosyncrasies of the individual and to gain a composite view representing the best thoughts of many men.

In the development of new ideas leading to the introduction of new methods, the societies could enlarge their present great field of usefulness if such discussion be had, and in my opinion further great good could be done if a number of men of different qualifications and experience would join in preparing a joint book, each dealing with his particular problem, but consulting the other men. In this way individualism would be preserved and various processes would be described, and if properly reviewed the paper would be palpably capable of analysis, and the student would have at his command the associated efforts of a group of qualified engineers.

I have neither the time nor the qualifications to make an investigation and study in detail of hydro-metallurgical processes, but for some years I have foreseen the development of such processes, and I have a deep interest in the subject and have given some of its phases careful consideration from a long-distance viewpoint. I have assisted in directing many experiments and in reviewing the merits of many processes, and am glad to make some introductory remarks with the understanding that other gentlemen present will freely and candidly enter into the discussion and help to fill out the few bare bones that I can uncover.

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You will see in my remarks that I am controlled very largely by operating demands. No matter how beautiful the process, simplicity and cheapness of operating costs are the main features that appeal to me, and complexity and increased costs only appeal to me if they definitely promise an increased yield that will more than offset the increased expense; for, given the grade, the maximum profit per ton is demanded.

As you all know, there has been a very large development of deposits of lean copper ores in the Southwest in recent years, and this region has rapidly developed until in the aggregate it produces more copper than any like area in the world, and it still has a large increase of output ahead of it. The vast mass of these lean deposits consists of sulphide ore, and until recently these sulphides alone have attracted serious attention. It is well known, however, that there are large tonnages of oxidized copper ore associated with the lean porphyries.

A few years ago the Calumet & Arizona Mining Co., upon the recommendation of J. C. Greenway, took an option upon a majority of the stock of the New Cornelia Copper Co., whose mining properties are at Ajo, in southwestern Arizona. Mr. Greenway directed systematic drilling upon this property, followed by the sinking of some 70 shallow shafts for test purposes and the sinking of two deeper shafts, with considerable underground development, for the purpose of testing and checking the grade of the sulphide ore. The stock was purchased.

The matrix of the ore is an eruptive granite. There is no overburden. The chalcocite has been oxidized nearly as fast as formed. The zone of transition from oxidized minerals to primary sulphides is narrow. The oxide and sulphide ore is remarkably uniform in grade. Ira B. Joralemon's paper, in the *Transactions*,¹ describes this deposit. He shows that there are about 40,000,000 tons of ore containing $1\frac{1}{2}$ per cent. copper, and the tonnage is divided into about 12,000,000 tons of $1\frac{1}{2}$ per cent. oxidized ore and 24,000,000 tons of sulphide ore, with about 4,000,000 tons of mixed oxide and sulphide between. If the grade be dropped to an average of less than $1\frac{1}{2}$ per cent., these tonnages would rapidly increase.

In parallel with this development, oxidized ore in notable tonnages has been found in the Globe mining district. This ore is more scattered, and, on the whole, is leaner than the Ajo ore, and frequently has a large capping of overburden. In the aggregate, the Inspiration Consolidated Copper Co. alone now has over 15,000,000 tons of oxidized ore running about 1.3 per cent., and a probable tonnage in addition. There are other mines in other districts containing oxidized ore, notably in Utah and Montana, and there are also oxidized ores at other points in Arizona and a vast deposit in Chile. The treatment of such tonnages is of prime importance.

¹*Trans.*, xlix, 593 to 609 (1914).

The Ajo ore has a matrix of granite with a large percentage of secondary quartz replacing feldspar. There is little calcium present in soluble form, and the oxidized ore seems to be best adapted to a leaching process. Preliminary laboratory tests showed that the ore crushed with the production of remarkably little slime and that the copper will dissolve in dilute sulphuric acid quite freely when the crushed ore contains fragments no larger than 6-mm. cubes. Tests also showed that a little cuprite is present, which is only partly soluble in sulphuric acid. There are also other soluble salts, notably salts of iron and aluminum. It was decided to investigate a leaching process for the oxidized ore, and since the ore made but little fines and could be leached with comparatively coarse crushing, we decided that we would try to leach in tanks by percolation. The amount of silver in the ore is small. Salt is apparently expensive, and it was decided that, other things being equal, the use of chlorine compounds as solvents was not desirable. I know, of course, that chlorine compounds may have a great future in leaching, but in this particular case, and in the early development of a process, it was considered that sulphuric acid offers fewer difficulties as a solvent.

To me the hydro-metallurgical problem divides itself naturally into two nearly separate divisions—the solution of the copper, and the recovery of the copper from solution. We were obliged, however, to study the problems of solution on the basis of getting the copper into solution in the form most convenient for its ultimate recovery.

At the start we employed Stuart Croasdale to undertake a study of the solution of the copper, and we also delegated to him experimentation upon the precipitation of copper by iron. Mr. Croasdale ably conducted a long series of most valuable experiments and published a paper upon this subject which you will find in the *Transactions* of the Institute.² At the same time we were fortunate enough to obtain the advice of Utley Wedge upon the manufacture of sulphuric acid direct from calcining-furnace gases.

Mr. Croasdale demonstrated that the ore crushed to an 8-mm. cube, or smaller, would yield over 80 per cent. of its copper when treated with a dilute solution of sulphuric acid. He demonstrated that slime presented no difficulty if the ore was distributed in the tanks uniformly, and that by finer crushing 85 per cent. of the copper could be obtained in solution if treated for a sufficient length of time with an acid solution. He demonstrated not only that the ore was permeable uniformly, but that with comparatively small tanks the height was not a matter of great consequence, and he was easily able to use a column 12 ft. in height, and could probably use one much higher. Later on we found upward percolation preferable.

Mr. Wedge showed us that with care in calcining the sulphide fines

² *Trans.*, xlix, 610 to 658 (1914).

for reverberatory furnaces at the Calumet & Arizona smelter at Douglas, sulphuric acid of 55° to 60° Bé. could be manufactured with a low operating cost, and a construction cost of not to exceed \$3,000 per ton of daily capacity. The point of manufacture is about 300 miles, more or less, according to the route, from the mine. Mr. Wedge also undertook a study of sulphating and chloridizing roasts. His requirements were that the ore be crushed to pass a 1½-mm. screen. He showed that if the sulphide ore could be mined in parallel with the oxidized ore, and in proportion to their respective tonnages, there would be sufficient sulphur for his purposes and that a very high extraction could be obtained. He also showed that sufficient advantage could not be had in using salt to offset its cost and the difficulties that might arise later. Mr. Wedge also showed that a small amount of heavy sulphides from the Bisbee mines crushed and mixed with the oxide ore would give a very high percentage of extraction after the mixture had been subjected to a sulphating roast. He also introduced the idea that a preliminary water leach would give a perfectly pure sulphate solution from which the copper could be recovered electrolytically and that the acid thus regenerated would be sufficient to complete the leach.

Frederick Laist in the meantime was conducting a number of experiments governing processes in a like brilliant way, and among these processes was the development of an oxidizing roast on Anaconda tailings. Like Mr. Wedge, Mr. Laist found that by calcining sulphides or mixed sulphides and oxidized ores a high percentage of the copper could be extracted by leaching. Temperature control was vastly important, as the developments of both Messrs. Wedge and Laist showed. If too high a temperature is used a large amount of the copper becomes insoluble in the strongest acid, but in both cases temperature control was demonstrated to be simple and practicable, and under these circumstances practically no insoluble copper compounds were formed. In Mr. Laist's case, however, the use of salt is essential on account of the silver contents of the charge, but he finds that it is not necessary to use this in a chloridizing roast and that a solution of salt with sulphuric acid gives him the desired extraction on the product from the oxidizing roast.

The manufacture of sulphuric acid by the chamber process, while simple, requires a very large investment, and in this case the transportation charges would amount to more than the cost of the acid. The chamber and the contact processes, of course, are the two important manufacturing processes, but even a casual study of wet methods of copper recovery shows that there are other methods of acid manufacture involving a principle similar to the chamber method, that might be used in parallel with the recovery of the copper from solution with less investment for plant and with a saving in energy that is important.

The McKay process includes a sulphating or an oxidizing roast, as

he chooses, to get sulphide of copper into soluble form, and he adopts the same process for this purpose as followed by Messrs. Wedge and Laist.

The Slater process and the Midland process deserve study. They depend upon the use of complex series of chlorides and hypochlorous compounds that will dissolve many salts of copper that sulphuric acid will not touch. They will attack, among other things, chalcocite, and they deserve serious consideration in many important but special cases, as they avoid the necessity of a preliminary roasting operation.

I regret that I am only slightly familiar with the very important work in the development of a process by E. A. Cappelen Smith for the beneficiating of the Chuquicamata ores.

RECOVERY OF COPPER FROM SOLUTION

At the beginning of our investigation we seriously considered what I called the "brutal method" of leaching, namely, the manufacture of sulphuric acid, the solution of the copper from the ore with such acid, and the precipitation of the copper by metallic iron, with the resultant complete, or nearly complete, destruction of both acid and iron and the production of an impure cement copper that will have to go through the process of smelting and refining. The results of Mr. Croasdale's tests and Mr. Wedge's cost figures indicated that such a course was commercial, and we concluded that the $1\frac{1}{2}$ per cent. Ajo ore, with no overburden, mined by steam shovel and crushed coarse, would yield a profit on a 12c. copper market if a cheap grade of Alabama pig iron were used.

But it occurred to me, as it had occurred to many engineers, that oxide of iron could be reduced to metallic iron without fusion, and if so we might bring Bisbee sulphides to the mine, calcine them for the manufacture of sulphuric acid and then metallize the available iron in the impure calcine, already containing values requiring recovery, and use this for a precipitant. I found at the start that metallic iron was being produced by this method with an inferior fuel from high-grade iron ore by an intermittent process; that is, by first heating to the desired temperature by internal combustion of a part of the excess coal in the charge and then allowing the heated charge to stand without the admission of air, but allowing the gases formed to escape freely.

After consultation with friends, we concluded to try the experiment of a multi-hearth furnace, and at first my idea was that possibly calcining could be done on the upper hearths and the hot calcine could then pass to the lower hearths, where it could be further heated and submitted to a reducing action. We erected a small Wedge furnace, but only tried to metallize, and not to calcine in it. Some of the iron oxide was reduced to metallic iron, but on the whole our experiment was a

failure because in keeping the charge hot the percentage of carbonic acid gas produced was sufficient to retard and reverse the action of carbon monoxide in reduction. In parallel with this, we also made experiments in revolving cylinders, aiming to reduce the iron to the metallic form with illuminating gas and with fuel oil by a continuous process. These also were failures.

Mr. Laist conducted similar experiments on a continuous process for making sponge iron at Anaconda, and ended with a like failure. He has, however, demonstrated that he can make sponge iron by an intermittent process in a revolving drum. It had become evident to me, even in my limited environment, that the simple use of iron, with the destruction of both iron and acid, to the production of an unfinished product, while apparently commercial, was in our extremely simple case preposterous, and investigation showed the splendid work done on other lines and convinced me that if considered from a commercial viewpoint better methods were bound to develop a process that would fit our requirements, and very much cheaper copper would result.

At the start we decided to investigate the electrolytic deposition of copper from solution, but a little later on we took up the subject of the direct precipitation of the copper by sulphurous acid gas. I prefer to attack the latter subject first for the purposes of this paper.

In the precipitation of copper by sulphurous acid gas, temperature and consequent pressure must be considered. If the copper is precipitated in metallic form this temperature and pressure are very high, but if a sub-chloride of copper would satisfy us, the temperature and pressure may be moderate. In both of these processes, the reactions we are familiar with in the formation of sulphurous acid by treating copper turnings with concentrated sulphuric acid are reversed by heat. For the production of metallic copper the action depends upon getting sulphurous acid dissolved in the sulphate solution in sufficient amount to precipitate the copper and then heating the mixture under proper control to about 150°C . The question of the use of an abnormally rich gas is involved. Under proper regulation pure copper is produced, which only needs melting for the market, and a large excess of sulphuric acid is formed. Mr. Van Arsdale has done valuable original work on this process, followed by the work of Weidlein and others in its commercial development. Laist has studied precipitating sub-chloride of copper. A soluble chloride is added to the sulphate solution of copper and the proper amount of sulphurous acid is added. Moderate heat under pressure— 90°C . of heat—precipitated almost all the copper as sub-chloride. The subsequent process of obtaining the copper without volatilization and the recovery of the chlorine offered difficulties. Messrs. Van Arsdale and Bacon have valuable experience on this subject.

These processes need further development and are most interesting.

It is with no disparagement of the process that we have abandoned it in our case. I simply feel that the method would be a complicated and an expensive one, and I fear working with heat and pressure where fuel is dear and a corrosive solution is used. To me and my friends the electrolytic method seemed best adapted to our condition and the simplest and more certain of commercial application.

Of course, the first difficulty was the solution of other oxides than copper and the danger of some loss of acid thereby, and, what was more important, the susceptibility of some of these dissolved neutral salts to oxidation by nascent oxygen to the production of acid salts, which in turn will dissolve copper from the cathode. It seems that such salts are most valuable as depolarizers, but must be held under control.

After investigating the subject I advised Mr. Greenway on behalf of his company, and we employed F. L. Antisell to make and direct a series of tests and experiments covering the precipitation of copper in the presence of iron and aluminum sulphates. For some time previous I had been familiar with experiments tried by Messrs. Pope and Hahn, and these gentlemen were employed to investigate in parallel the practicability of removing enough of the iron and alumina from the solutions as they foul to keep them below the danger point in electrolytic work. Both Mr. Antisell and Messrs. Pope and Hahn began their experiments upon a laboratory scale at the Raritan Copper Works. Messrs. Pope and Hahn found the leaching could be so conducted that the rate of increase of iron and alumina was slow. They therefore decided to leach in a certain way and to treat a certain high iron and aluminum neutral solution with copper oxides by agitating, heating, and injecting air. This is a rejuvenation of the old Hoffman process applied to a new purpose. The iron and aluminum sulphates in such foul solution, together with arsenic and other impurities, are precipitated as insoluble oxides and the acid previously combined with these salts is taken over by the copper oxide and forms sulphate of copper. The clarified solution, with these impurities removed, is then mixed with the balance of the solution, thus keeping an iron content of, say, less than $\frac{3}{4}$ per cent.

Mr. Antisell began his experiments by using a patent anode. This anode was a narrow, rectangular wooden or lead frame with thin wooden sheets as diaphragms, forming a long and deep but very narrow box. It was packed with a clean coke in lumps crushed to pass a 1-in. ring, and contact was given by carbon rods imbedded in the coke. A solution of sulphate of iron and aluminum in great excess was made up and sulphuric acid added. This was used as a solvent for the copper. After the solution came from the ore it was at first passed to an absorption tower, where sulphurous acid gas was absorbed, and then it passed to the electrolytic tank. I go into no detail in describing Mr. Antisell's flow sheet and his process, as he himself is present. After promising results had

been had in the laboratory tests at Raritan showing the possibility of high extraction and low power costs, Mr. Antisell sent Mr. Jamieson to Douglas, where tests were made with full-sized anodes. Excellent results were obtained here, but difficulty was had with the complete absorption of sulphurous acid. Also, the anode was cumbersome and not adapted to operating conditions, but, notwithstanding, excellent results were obtained experimentally and he ran for a long time with a recovery of nearly 2 lb. of copper per kilowatt-hour, an extraction of 80 per cent. of the copper contents of the ore, and the production of nearly the theoretical amount of sulphuric acid from the SO_2 gas.

Messrs. Pope and Hahn later went to Douglas and began experiments, and also later conducted a most valuable set of experiments on the manufacture of acid-proof coatings for concrete that will neither crack nor soften under the high temperatures of the Arizona desert, and which will adhere closely to the sides of the tank. This work is very important and needs mention.

Mr. Greenway discovered that if a certain copper solution containing salts of triad elements is treated in a simple way, the iron or aluminum triad salts may be removed. Dyad salts are not affected. He was led to the belief that such oxides thus precipitated were in large part insoluble in dilute acid and that therefore he had a very simple method of removing iron and aluminum from solution. Against my prediction, laboratory tests seemed to show his contention was correct. Having used up about 300 or 400 tons of Ajo ore at Douglas for the Croasdale, Antisell, and Pope and Hahn experiments, and having promising results, the engineers moved to the mine at Ajo. Mr. Greenway erected a small plant of his own, and a Pope and Hahn plant was installed and the use of sulphurous acid was temporarily discontinued. As for the Greenway process, I am not yet at liberty to discuss it further. But I may say over 100 charges were run with this process. The iron content in the solution was kept down to less than 1 per cent., and we obtained an average of slightly over 1 lb. of copper per kilowatt-hour, but the extraction was low. We then extended the time of the leach and obtained a higher extraction, and finally we began to crush the ore, which at that time had been crushed only to $\frac{1}{2}$ -in. ring, finer, and found that if crushed to pass $\frac{1}{4}$ -in. mesh percolation was not retarded and an extraction of over 80 per cent. was practicable. The iron has reached as high as 1.2 per cent., but we are still getting a pound of copper per kilowatt-hour from the electrolytic tanks.

These experiments have been so successful that we are proceeding with the erection of a 40-ton plant, where tanks will be used that will hold 60 tons, if necessary, and will be abundantly large to permit of a study of the comparatively even distribution of crushed ore, and we shall proceed with these most interesting experiments, using first, as we have

in the smaller tanks, the Greenway process. In the meantime, the 1-ton plant will be liberated and we will immediately proceed with the continuance of a modified form of the Antisell system of leaching. He finds that with the construction of better absorption towers we can absorb the necessary amount of SO_2 gas, and he has demonstrated that with the use of SO_2 gas, using dyad iron as a depolarizer and triad iron as an oxidizer, he can manufacture sulphuric acid. We can, if successful, obtain $1\frac{1}{2}$ lb., and possibly more, of copper per kilowatt-hour, and at the same time manufacture sufficient acid to replace wastage. The electrolytic copper in all cases has been fully up to the trade standards.

CONCLUSIONS

My deductions from a study of the work of the able engineers engaged in these extensive operations are as follows:

1. Dilute sulphuric acid will give a very high extraction on oxidized ores of the Ajo type. Time is a more important element in extraction than strength of acid. A maximum size of 6-mm. cube is required. Such ore crushed to the maximum size permits percolation and the production of a clear solution because the slime content in the ore so crushed is very small.

2. Where such ore is mined at a cost of between 25c. and 50c. a ton, it is commercial on a 12c. market, even if leached with sulphuric acid and the copper precipitated with pig iron to the waste of the acid and iron and the production of an impure cement copper that requires further treatment. Metallized iron from calcines can be made and will prove a little cheaper than pig iron if it is marginal.

3. An electrolytic method for the recovery of pure copper from sulphate solutions is far preferable and will produce cheap copper. In all cases a low current density is required. Hard-lead anodes or composite anodes of hard lead and coke of uniform thickness will be used.

4. Such electrolytic method, with control of the amount of iron and aluminum in the electrolyte, and without the use of sulphurous acid gas, is practicable. An acid loss of about 1.5 lb. per pound of copper will be sustained. Chamber acid will be required.

5. I believe there is a decided probability that the cheapest and best method will be the electrolytic deposition from sulphate solutions without iron control, and the use of the requisite quantity of sulphurous acid. The sulphurous acid will be absorbed without the tank house and in quantity that will not cause annoyance. Iron and aluminum will act both as depolarizers and catalytes. This process promises a high yield of cathode copper per unit of power and the manufacture of sufficient sulphuric acid to replace the waste.

6. While these remarks apply to the Ajo oxidized ore and ores physically and chemically similar, they need not apply to ores of a different composition. Mixed oxide and sulphide ores, be the sulphide chalcocite

or chalcopyrite and bornite; ores containing great quantities of clay, or presenting other difficulties, have to be considered separately, and there is beyond question a great field for other processes and for other methods, both for obtaining the copper in solution and obtaining the copper from the solution.

And now, gentlemen, while thanking you for your attention, I end my theme as I began. Notwithstanding the importance of this subject, it covers but a corner in the metallurgy of one metal. There are many other subjects which if discussed in committee and with an open mind will lead to the advance of ourselves and of our institutions.

DISCUSSION*

UTLEY WEDGE, Ardmore, Pa.—It has certainly been very gratifying to hear such a comprehensive discussion of the development of the technique of leaching in connection with the property of the New Cornelia Copper Co. The presentation of this matter in the paper as read I believe to be accurate, but one aspect of the problem has not been presented. The paper relates almost exclusively to the treatment of carbonate ore. The deposit at Ajo, where this method is to be applied, comprises about twice as much sulphide ore as carbonate ore. It also comprises 1,500,000 tons of ore which is a mixture of sulphide and carbonate ore. It would be interesting to have accurate data as to the yield or percentage recovery of copper that can be secured from the sulphide ore by the various methods of concentration. The character of the ore leads me to believe that the percentage of recovery of values from the sulphide ore by any of the wet concentration or flotation processes in use at the present time, will be materially less than the percentages which this paper has shown us it is possible to recover by a leaching process. In considering the ultimate beneficiation of this property, I would suggest that the treatment of the carbonate of copper ore should always be considered in connection with the treatment of the balance of the deposit.

Is it not true that the sulphuric acid leaching plant for carbonate ore as described, if constructed on a large scale, will have largely outlived its usefulness when one-third of the deposit of ore has been worked out? If the same facilities that will handle the carbonate ore would also handle the sulphide ore, so that the same plant and the same processes could be utilized throughout the operation, the eventual total cost of construction might be materially reduced. The leaching of carbonate ore of this character with sulphuric acid is clearly demonstrated to be a success. It is clearly demonstrated that the carbonate ore which is to be leached with sulphuric acid does not need to be finely ground to leach completely and economically. It is apparently demonstrated that with processes recently perfected SO_2 can be introduced into an impure

* Presented at a meeting of the New York Section, Jan. 6, 1915.

electrolyte with advantages in economy as regards consumption of power in electrolysis and quality of product, but these advantages can largely be retained by the following operation:

Let the sulphide ore and the ore which is found between the mass of carbonate and the mass of sulphide ore, which partakes of the character of both, be ground to 20 or 30 mesh and furnaced at a temperature not much above a red heat, and the material can at once be leached with as good percentages of recovery as have been secured in the case of the carbonate ore. The copper solutions secured can be electrolyzed, and sufficient sulphuric acid can be produced from the electrolysis for use in leaching all or a portion of the carbonate ore, which methods can be identical with the methods described in the paper.

The leaching of the finely ground sulphide ore will call for somewhat modified appliances on account of the fineness of the material, but it has been abundantly demonstrated in other plants and in other camps that finely ground material can be economically leached. There are methods of handling finely ground material in leaching which even seem to have advantages in operation as compared with the methods described for leaching material in lumps.

In case the leaching plant projected, as indicated by the paper read, is built no larger in capacity than would in any event be necessary in connection with the treatment of sulphide ore and carbonate ore by the methods indicated, then and to that extent the construction for such acid leaching of the lump material would be serviceable for the life of the property, if the construction is of such character as to last that long, and if the mining and beneficiation of the carbonate ore can be so arranged that it will extend over a like period.

I am of the opinion that sulphatizing roasting of the sulphide ore in the orebody of the New Cornelia Copper Co., coupled with the acid leaching of all or a portion of the carbonate orebody with sulphuric acid by the methods indicated in the paper, will give the lowest possible cost of operation and will give the highest possible recovery of values of any method known to the metallurgical art. This combination of methods would obviate the necessity of construction of a sulphuric acid plant; in the place of this plant there would be constructed a furnace plant for giving the required roast to the sulphide ore. At the start, with only carbonate ore accessible, enough of the carbonate ore could be roasted mixed with small percentages of iron pyrite from neighboring camps to sulphatize the carbonate of copper.

I believe the cost of a roasting plant for conducting the process in this way is amply justified by the large recovery of values which can be made, and I believe that the advantages of this combination of methods are so great that the beneficiation of this property by such methods would yield several million dollars in excess of what would be secured if the sul-

phide ore were treated by any of the known concentrating processes followed by the smelting of the concentrate.

As a general statement, the class of copper ores known as oxidized ores and carbonate ores, to the extent that they are free from alkaline earths or other gangue materials that are readily attacked by sulphuric acid, will, in general, lend themselves to the treatment that Dr. Ricketts has described: namely, the treatment with sulphuric acid. But the past history of the leaching method as applied to the metallurgy of copper is full of instances where efforts have been made to accomplish results by leaching processes not preceded by the study which was given to this matter in connection with the New Cornelia operation. Such attempts in the past have justified the criticism of leaching processes which was once made in a general way by the father of a gentleman whom I see present, who said that the chief thing he had against leaching processes was that they were so *Eine Schweinerei*. If I am asked to give the English equivalent of that, I would say that it is apparent that the installation of leaching processes which the German gentleman had seen did not appeal to his esthetic sense.

But another day is very rapidly approaching, and the advance that has been made in the past few years in this line of progress seems to me to be revolutionary. There is so much being done by so many different companies that the efforts of any individual in connection with such practice must seem small.

The connection that I have had with this matter had led me to emphasize especially the effort to secure pure solutions which would yield to electrolysis without difficulty; and the development which has been brought about by two gentlemen who will be heard from this evening makes it seem probable that solutions quite impure in character will hereafter be economically electrolyzed. I regard the work of these gentlemen as of the utmost importance. The application of leaching processes to copper ores and copper calcines is practiced on a larger scale than is generally known. Over 1,000,000 tons yearly are treated for the extraction of copper by leaching. The largest leaching operation in the world is one operation which treats about 750,000 tons per annum. I refer to the Rio Tinto Co. The process used by the Rio Tinto Co. is in its essential character a sulphatizing roasting process. The ore is piled up in heaps 30 ft. deep, which are called *terreras*, and the ore is allowed to heat by chemical action until a sulphatizing roasting condition is produced and sulphates of copper are formed, which are leached out. The balance of the leaching of copper ores, done on a large scale throughout the world, is mostly chloridizing leaching, where salt is used. The leaching process with salt is very extensively carried out in Germany and England. There are several hundred thousand tons, I think, chloridized every year in Germany and England. I know individually

of quite a number of very large operations there with which I have come in contact in connection with the installation of furnaces for chloridizing roasting. There is a very considerable tonnage chloridized in this country. The chloridizing process is familiar to a great many here. For the benefit of the others, I will say that in general, ore, which has previously been roasted to deprive it of most of its sulphur, down to a point where the sulphur is from one and one-quarter to one and one-half times the copper content, is mixed with a percentage of salt, usually in good practice in the neighborhood of 10 per cent., and this mixture, which is reduced to a fineness of 3 mm. by grinding, is then heated in chloridizing furnaces to a dull red temperature, or even slightly below, and the copper is chloridized and subsequently leached out with water, and with such hydrochloric acid as is caught in the scrubbing towers in connection with the furnaces.

The use of sulphur dioxide in connection with the precipitation of copper is a matter which personally I have been watching with very great interest. The quantity of sulphur gas which is escaping from the different smelter plants in this country is so prodigious that it seems like a waste; it seems as if this material will eventually be utilized and turned to some account. It is very possible that to some extent this will be brought about in connection with either the electrolytic or other precipitation of copper, where sulphur dioxide is evidently going to be very much in demand. There has been a possibility, if not a probability, that liquid SO_2 might be required in this connection. It is interesting to note that at the present time, as far as I know, there is no liquid SO_2 plant in the United States. There are several in Germany, and one or two in England, and the stoppage of the importation of liquid SO_2 from Germany has advanced the price to about 10c. a pound at the present time. This certainly does not look attractive as a metallurgical proposition, but the possibilities of the cheap production of liquid SO_2 to permit of its being shipped considerable distances for use, are such that I think it will come to pass in this country in a comparatively short time.

E. A. CAPPELEN SMITH, New York, N. Y.—Our plant is practically finished; the leaching plant itself, as well as the mining plant, is practically completed. We have been slightly delayed with the completion of the power plant on account of the war in Europe—our power plant is being manufactured in Germany; according to our present plans, we expect to have the plant in operation by the first of April. So far in the construction work everything has gone on very nicely, and it now remains only to start the plant.

The general plan of work consists in steam-shovel mining the copper deposit followed by leaching the ore. From the crushing plant the ore will be moved by a belt and unloaded by a bridge that straddles the leaching tanks. The leaching tanks—six large concrete vats set end to

end—are each 160 ft. long, 110 ft. wide, and 16 ft. deep. Each tank is designed to hold 10,000 tons of ore. After the ore has been delivered to these by the traveling bridge from the main belt, the first solution is applied, by upward displacement. At first we had planned to conduct the operation entirely by downward percolation, but we found a slight tendency of the finer material to clog the filter bottom; also, we found that channels were formed in the charge. These difficulties were entirely overcome when we instituted the method of applying the liquor by upward displacement. The first treatment solution stands on the ore for 48 hr. This solution is immediately followed by the washing liquors in rotation, run on the top of the ore, and displacing the previous solution by the piston method. In this way, we find that we can wash the ore so that the tailing will contain only from 0.015 to 0.03 per cent. soluble copper. We think this is about the limit, and we can obtain this result without introducing any more water in the process than sufficient to make up for that carried off in the tailing as moisture, and in the solution which is discarded to get rid of the surplus acid produced from the ore.

The solution is removed from the tanks through lead-lined iron pipes, and conducted to a central pumping station, where we will use centrifugal lead-lined pumps, each with a capacity of 5,500 gal. per minute against a lifting height of 62 ft. The pumps have been built by the Worthington company; they have been tested in the manufacturers' shops, and as far as we can tell they are going to be highly satisfactory.

The solution is then delivered to the storage tanks, placed at an elevation above the leaching tanks, and from there it flows by gravity to the plant where we are going to eliminate the chlorine. We propose to pass the solution through cylinders 30 ft. long and 4 ft. in diameter, half filled with shot copper. By this means, the chlorine is precipitated in the form of cuprous chloride. The operation is continuous. The mixture of precipitated cuprous chloride, with the copper sulphate solution containing about 3 per cent. sulphuric acid, is run into Dorr thickeners, where the precipitate is separated from the solution. The cuprous chloride pulp corresponds to between 1 and 2 per cent. of the total amount of the solution. We are installing some Kelly filter presses, which are lead lined, and we are also going to install an Oliver filter, likewise lead lined, in order to compare the two types.

The filter-pressed material will be delivered to the smelter by the usual methods, and at the smelter it will be mixed with limestone and coke finely crushed (8- to 10-mesh material), put through a pug mill, pressed into briquets, and delivered to the blast furnace. In the blast furnace will be produced copper, and calcium chloride as a slag. The copper will be taken to a small reverberatory furnace, where it will be granulated and again used in the dechlorinators. We find that on account of the oxidizing agents present in the solution, partly in the form

of ferric sulphate and partly in the form of nitric acid, the efficiency of the copper shot is only about 50 per cent. of the theoretical; in other words, there is a direct solvent action on the metallic copper in addition to that caused by the reduction of the cuprous chloride. Practically all of the copper produced in the form of cuprous chloride will go back into the process.

The solution, after the chlorine is removed, is fed by gravity into the electrolytic tanks with a strength of approximately 5 per cent. copper. It will run through the depositing tanks, arranged in sections of 16 tanks each, which will have about 13,000 amperes to a tank. The voltage will be about 2.3. We expect a yield of approximately 1 lb. of copper per kilowatt-hour. The solution will leave the last tank containing about 1.5 per cent. copper. We find that the current efficiency remains high down to about 1.5 per cent. copper; if we go lower than that, the current efficiency will drop off rapidly. By holding to these limits we will have an ampere efficiency of approximately 90 per cent. The solution from the last tank in the cascade will be delivered back to the leaching plant to be used over again.

Our problem is somewhat unusual in this respect, that we have available sulphuric acid present in our mineral, and instead of having to look around for means of producing acid, we have to look for a method of getting rid of the surplus sulphuric acid. We will have to dispose of a quantity of the solution on account of its content of sulphuric acid.

There are no soluble impurities in the ore itself, such as antimony and arsenic. Soluble lime and alumina and alkali salts are negligible. Consequently, the usual trouble experienced in connection with leaching processes is not present in our problem. Chlorine, which we have in the upper part of the orebody, was really our most annoying difficulty, but the treatment of the solution with metallic copper has eliminated this difficulty. In addition to this, we also have a small amount of nitric acid in the form of nitrates. So far we have not been able to determine that any especial difficulty arises in the process itself from the presence of nitric acid. It has, however, a rather deleterious effect on lead connections, lead pipes, and lead anodes. We have noticed, however, that this action occurs only at points where we are handling the strong solutions—those containing 8, 9, or 10 per cent. sulphuric acid. Nitric acid is present up to 0.8 per cent., but outside of the difficulty mentioned, it does not give any trouble. Its presence does not seem to have any particular function in the process itself, and we have not been able to determine that its being there causes any lower current efficiency. It does not seem to have any effect on the voltage.

Chlorine seems to have a tendency, which was to be expected, of lumping the deposit to a certain extent, and the use of colloids to prevent this does not seem to be very encouraging. I think it will be a question,

more than anything else, of watching the deposit, and perhaps not allowing the cathode to get quite as thick as in ordinary practice.

Underlying the large oxidized orebody, at present developed to the extent of 200,000,000 tons, we have a sulphide orebody. Small-scale tests conducted on this material show that it can be treated either by roasting and leaching, or by water concentration followed by smelting in the usual manner. This orebody is also of enormous extent, but details have not as yet been decided upon for the final working of it, as the 200,000,000 tons of oxidized ore already developed will keep us busy for some time to come.

GEORGE D. VAN ARSDALE, New York, N. Y.—During the last few years we have been doing considerable experimenting on methods of precipitating copper. The first patent on precipitating from sulphate solutions by heating with SO_2 under pressure was taken out by us (U. S. Pat. 723,949).

This reaction, $\text{CuSO}_4 + \text{SO}_2 + 2\text{H}_2\text{O} = \text{Cu} + 2\text{H}_2\text{SO}_4$, is interesting, but there are a number of practical difficulties which will be met with in carrying it out on a scale of any size, the main ones being that it is not easy to get a sufficiently strong solution of SO_2 to carry out the reaction, and that the design of an apparatus for heating large amounts of corrosive solutions under pressure is a considerable problem. If these can be overcome, I believe the process may be developed, but I agree with Dr. Ricketts' opinion that electrolytic methods of precipitation are preferable.

Regarding our work at Douglas, we have felt that to say much about it was a little premature, as experiments are still in progress, but some notes on the preliminary experiments may be of some interest.

The general method which we adopted for the tests was sulphatizing roasting, leaching, and electrolysis from sulphate solutions, using depolarization.

The reason for the consideration of the sulphatizing roasting was that the materials which we have to treat contain in most cases large amounts of soluble alumina, and therefore, with a regenerative process only, we would be almost sure to have a deficiency of acid. With the sulphatizing roasting this deficiency of acid would be more or less made up.

The plant used for the preliminary experiments consisted of a 20-ft. six-hearth Wedge muffle furnace, several small leaching tanks, and an electrolytic installation designed to precipitate about 250 lb. of copper a day.

The objects of the roasting experiments were to determine the capacity of the furnace, the fuel consumption, and the percentage of extraction, both water soluble and acid soluble. The objects of the electrolytic experiments were to determine the power and the ampere efficiency, to determine whether the anodes used in our preliminary experiments

would be satisfactory under working conditions, and also to determine the best methods of dissolving and applying the SO_2 gas.

The results of our roasting experiments were as follows: We roasted three principal classes of material, in which the average insoluble copper in the calcines was 0.11, 0.16, and 0.41. The water-soluble copper showed 70 per cent. of the copper contents, 40 per cent., and 51 per cent., respectively, and the total extraction was 92.9, 88.0, and 70 per cent. One of the principal results of these experiments was to show us that for this class of roasting it was necessary to have more or less fine crushing as a preliminary.

The results of our preliminary electrolytic experiments were fairly satisfactory. We obtained a total amount of copper precipitated of something over 2 tons at an average figure for power of about 1.6 lb. of copper per kilowatt-hour. We produced acid per kilowatt-hour equivalent to about 5.1 lb. The ampere efficiency was low. This, however, was due to several reasons not directly connected with the process. We had a large number of short-circuits and other troubles, but we felt quite sure at the end of these preliminary experiments that we should get about 2 lb. of copper per kilowatt-hour. We had, however, a certain number of uncertainties and some anomalous results, so we started some further experiments on details of solution and other matters.

These further experiments resulted in the following general conclusions:

First, that it was not necessary to have such a large amount of SO_2 in solution as we had previously thought, and that, in addition, much better results could be obtained by using SO_2 together with iron, rather than either one alone. In other words, the function of the SO_2 was not only to act as a depolarizer, but also to act chemically in keeping the iron reduced. We also found there was a certain favorable effect from a high aluminum content. The temperature increase is favorable in the same way as in copper refining. With iron salts present there is a decreased ampere efficiency with increased temperature. This was counteracted by SO_2 and by increased current density.

As a result of these further experiments, we laid down the conditions of high ferrous sulphate in solution, high alumina, a temperature of about 115° to 130° F., current density of about 15 amperes per square inch, and sufficient sulphur dioxide present to reduce the voltage, and maintained sufficiently to keep the iron in the ferrous condition, and under these conditions I expect it will be possible to get a yield of from 2 to 2.5 lb. of copper per kilowatt-hour.

As to "foul solutions," which has heretofore seemed to be the principal bugbear in electrolysis, as long as the iron can be kept in the ferrous state, which can be done by SO_2 , it is fairly safe to say that, up to certain limits, better results will be obtained from so-called foul solutions than from pure copper sulphate solution.

We are now about to start a 75-ton plant, in which we expect to apply the information which we have gained to the treatment of one class of ore, and the result of the work which will be done in that plant will no doubt be published later.

RAYMOND F. BACON, Pittsburgh, Pa.—We have been in the field of copper investigation for some time, as many others have, but we feel it is somewhat premature to say very much as yet. We have been working on several different lines, and we believe no one leaching process will be developed that would be applicable to all ores or all conditions. There are certain leaching processes which are better adapted to the conditions of certain ores, and other processes are better adapted to the conditions of other ores.

As regards electrolytic processes, I think a good deal of the work we have been doing has been pretty well covered in different ways by the statements which have been made here to-night. Our work on electrolytic processes has been along two lines. First, to ascertain the optimum conditions for the electrolytic precipitation of copper from sulphate solutions obtained by the leaching of oxidized copper ores. In this phase of the work I may say that we finally obtained conditions under which we ran a solution over the ore through the precipitating tanks and then over the ore again, 20 cycles being completed, without any purification of the solution. There was precipitated on the average in these 20 cycles 1.86 lb. of copper per kilowatt-hour and the current efficiency on the twentieth cycle was fully as good as that on the first cycle. Our second line of work on electrolytic processes has been an attempt to find efficient and cheap depolarizers, so as to increase the amount of copper precipitated by a definite amount of electrical energy. We have found one very cheap substance which for short runs gave us 4 lb. copper per kilowatt-hour.

In regard to the sulphur dioxide process, we have been actuated by this thought—it is necessary in most ores to have quite an excess of acid to take care of mechanical losses and to take care of alkaline substances in the ore. One way of doing this is along the line which has been spoken of, by using the sulphur dioxide in an electrolytic cell. We, also, have used sulphur dioxide in an electrolytic cell, but all of our work has tended to show that the efficiency of sulphur dioxide as an anode depolarizer in a sulphate solution is rather low. Because of this low efficiency, our calculations showed us that it is cheaper to precipitate directly with the sulphur dioxide under pressure than to use the sulphur dioxide as an aid to electrical precipitation; that is, the heat required for direct precipitation with sulphur dioxide will be cheaper in most localities than the electrical energy required either for straight electrical precipitation or for the combination in which sulphur dioxide assists the current. We all appreciate the advantage which any electrolytic process has of giving directly cathodes

of electrolytic copper. On the other hand, it has seemed to us that for most localities and most conditions the disadvantages of electrolytic processes, as at present developed, outweigh this advantage. Some of the disadvantages of electrolytic processes are these: The fouling of solutions immediately and seriously cuts down the current efficiency. Changes in the character of the ore, particularly as regards soluble impurities, are always to be expected in large orebodies. The electrolytic process is very sensitive to changes thus brought about in the electrolyte. The installation and upkeep costs of the electrolytic precipitation process are very high and to obtain efficiency requires constant supervision by high-priced men. One rather fundamental objection to electrolytic processes has not been discussed here to-night. In electrolytic processes, at least of the type which have been spoken of to-night, the copper content of the electrolyte can be reduced in the precipitation cells no lower than 1.5 per cent. copper. That means that it is not easy to so conduct a cycle of leaching, precipitation, etc., that all of the water-soluble copper shall be removed from the ore and the volume of the solution in the cycle shall be kept constant. For example, let us assume in a systematic leaching process that all the copper has been extracted from a ton of ore and that that ore is then in contact with the solution which in the cycle is strongest in acid, having just come from the precipitation tanks, and which, of necessity, still contains 1.5 per cent. of copper. This one ton of ore is drained as thoroughly as possible and, according to our experience, then contains from 40 to 100 gal. of this solution containing 1.5 per cent. copper. In other words, there is from 6 to $13\frac{1}{2}$ lb. of copper still in this ore. By adding the proper amount of wash water to retain the volume of solution in the cycle constant—that is, in this particular case from 40 to 100 gal. of wash water—the copper content of that ore will be reduced to from 3 to about 7 lb. To wash out further this soluble copper, so as to have a negligible quantity of soluble copper remaining in the ore, according to our experience, requires an amount of wash water such as to increase very appreciably the total volume of solution which is in the cycle. It is obvious that in any process in which there is practically complete precipitation of the copper, as in Weidlein's sulphur dioxide process, this washing difficulty does not arise. In place of losing soluble copper in the ore, one there loses a corresponding amount of sulphuric acid; and in Weidlein's process sufficient sulphuric acid is constantly being formed, so that this loss can be taken care of. As to the element of danger in processes in which a solution is heated under pressure, I would call your attention to the fact that there are thousands of boilers operating continuously in the United States under pressures considerably in excess of those used in the Weidlein process. We built in Nevada a 40-ton plant and we have had certain mechanical difficulties, as we expected to have. These difficulties we are meeting by the installation of

a continuous process of precipitation. The advantages of continuous precipitation will be evident to all mill men. Our expectation that the cost of heat in this process of precipitating pure copper is less than the cost of electricity for electrical precipitation has been borne out. Of course, I recognize that there are localities where electrical power is so cheap that an electrolytic precipitation process should be the one to be considered, but I believe that for most localities in the western United States some of these other processes are more favorable as regards cost. In the Weidlein process, a very large excess of acid is constantly regenerated and all who have had experience in the leaching of ores know how desirable this is. In that connection I will say that we have developed a system of handling sulphur dioxide and a method of concentrating it from dilute flue gases. We believe we have made noteworthy progress in that direction, but I am not prepared at this time to go any further into that. In a short time we will be able to say a little more on the subject.

FREDERICK J. POPE, New Rochelle, N. Y.—The investigations in connection with the leaching of copper ores to which I will refer were carried on by A. W. Hahn and myself. In our preliminary work we were assisted by Messrs. Bryan and Aldrich of the Raritan Copper Works.

In our first considerations of the work, we had to decide: The solvent we would use; how we would precipitate the copper from the solution, and what we would do with the iron in the electrolyte. Without going into the details of the reasons, we decided that sulphuric acid would be the most satisfactory solvent and that we would recover the copper from the solutions by electro-deposition. As to how we would take care of the iron in the electrolyte, we had to choose between controlling the iron (*i.e.*, keeping it in the ferrous state) in the solution, and eliminating it from the solution. The removal of the iron appeared to offer the more attractive field.

Briefly, the method followed involves leaching with sulphuric acid, removing a portion of the iron from the solution, and precipitating the copper electrolytically.

Regarding the leaching *per se*, I do not think I could say anything which would be of any particular interest. There are certain factors which govern the extraction of copper from an ore, and when these factors are given due consideration, I think we can get about the same extraction with one process as with another.

In considering how to remove iron from the electrolyte, we were reminded of the work which had been done by Hoffman at Argentine, where he precipitated iron from bluestone solution. Hoffman accomplished this by heating the solution in a suitable tank, agitating with air, and adding copper in an oxidized form. I think he used a roasted lead-copper matte. This work of Hoffman's gave us our idea for the removal of iron from the solution.

For the copper oxide necessary in this work we have used high-grade copper carbonate ore, oxide formed by the roasting of cement copper, roasted high-grade sulphide ore, also roasted copper sulphide concentrates. In our demonstration plant (10 tons daily capacity of carbonate ores) we used roasted concentrates. The concentrates were roasted in a small Wedge furnace, the results obtained being far better than those attained in our preliminary work when we roasted the same class of concentrates in a laboratory muffle furnace.

In order to obtain a satisfactory reaction between the copper oxide and the iron, etc., in the solution, it is imperative to have the oxide material very finely divided. The roasted concentrates we used were ground in a small Abbé mill. We found that with 5 or 6 hr. grinding we obtained a product 93 per cent. of which would go through a 200-mesh screen. It is probable if we had had time to investigate this grinding operation, we would have found that it was not necessary to grind for 5 hr., but, owing to the shortness of the period at our disposal for securing results, we did not have the opportunity to bring this operation down to the shortest time limit.

From the Ajo carbonate ore we removed a little more than 2 lb. of iron per ton of ore treated. In order to keep the iron in the solution below the percentage deleterious to electrical efficiency, it was necessary to treat approximately one-quarter of the solution. We found that by manipulating the solution going on and off the ore in a certain way we could concentrate a large proportion of the iron in a certain part of the solution. This was accomplished as follows: The first solution going on the ore carried about 2.5 per cent. of free acid and was kept on the ore by circulating until it was approximately neutral. This neutralizing precipitated from the solution on the ore a considerable proportion of the iron, alumina, etc. This neutral solution was drawn from the ore, and after acidification by electrolyte coming off the electrolytic tanks was sent to electrolytic tanks for precipitation of the copper. As the first solution was withdrawn from the leach it was replaced by fresh solution. This fresh acid solution not only removed some more of the iron from the ore but also picked up the iron which had been precipitated from the first solution used. This second solution now high in iron was circulated until it was nearly neutral, when it was replaced by fresh solution carrying 4 to 4.5 per cent. free acid, which completed the leach. The total leaching period averaged about 80 hr. The solution high in iron, etc., was sent to that part of the plant where removal of iron and other impurities was accomplished.

In purifying the solution it was first heated to 195° to 200° F. When this temperature was attained agitation with air was begun and the finely ground oxide material was gradually added. Maintenance of a temperature of about 195° F. and agitation with air was continued for

3.5 hr. During the first hour all the ferric iron was precipitated. The removal of the ferrous iron was slower and we found that it was not advisable to endeavor to precipitate it all. In 3.5 hr. we easily precipitated 90 per cent. of the total iron and at the same time removed 65 to 75 per cent. of the alumina, 50 per cent. of the manganese, and all of the arsenic, antimony, and bismuth which might be present.

After the above operation the pulp was passed through a Schreiber wooden filter press. The solution came out beautifully clear and was sent to the high-acid electrolytic circulation system. We anticipated that with the precipitate produced we might have trouble with the filter cake; that it would be slimy and difficult to handle. We did not find it so. The cakes were 2 in. thick, firm, washed well, and the amount of copper in the form of unused oxide they contained was well within commercial limits.

As you are well aware, when the iron sulphate, aluminum sulphate, etc., react in the agitation tank with the copper oxide, they give up their acid radicals to the copper, forming copper sulphate, which in the electrolytic tanks yields sulphuric acid, so that the only loss of acid is that due to combination with the alkalies and that due to loss by entrainment in the tailing. This acid loss we found we could easily replace by a proper control of the roasting of the concentrates in the Wedge furnace; *i.e.*, we could so control the roast as to maintain in the calcine sufficient copper sulphate either as water-soluble sulphate or as basic sulphate to compensate for the acid losses to which I have referred.

Regarding the electrolytic part of the work, we simply followed standard tank-house liberator-tank practice. Our anodes were antimonial lead, 4.0 per cent. antimony. We tried both grids and sheets. With the grids our voltage was 0.3 volt higher than with the sheet anode and we discarded the grids in favor of the sheets.

With reference to the use of a lead anode, I might say we commenced our work with some apprehension, fearing that there might be a serious loss of lead. We made a number of protracted tests and as a result we calculate that the cost for lead anodes at Ajo with lead at 8c. per pound would be less than 0.2c. per ton of ore treated.

In the process I have outlined there is really nothing new or novel. It simply consists of an idea picked up here and another there, and the putting of these various ideas together.

If we were to criticise the process, the pros and cons might be summed up about as follows:

Advantages:

a. Simplicity.

b. Every step in the process has been in use in commercial plants for years.

- c. Saturated solutions avoided.
- d. Wash water not excessive, and if it should become so, it can be taken care of.
- e. Impurities which would injure the quality of the electrolytic copper are eliminated from the electrolyte.
- f. Acid control very simple.
- g. No step in the process is dependent upon fine adjustments.

Disadvantages:

- a. High and low acids used. This necessitates more solution-storage tanks, more solution lines, and more pumps than if a single solution were used.
- b. There is an extra operation: namely, purification of electrolyte, which involves the installation of a roasting furnace, boilers, compressor, and filter press.
- c. The lead anodes will probably have to be changed at the end of four or five months, since the lead peroxide scale formed will increase the voltage.

J. PARKE CHANNING, New York, N. Y.—The problem presented in the Miami district is a little different from that in the districts referred to in the discussion of this evening, inasmuch as the ore to be treated there is one which is a mixed chalcocite and carbonate. The gangue, as you know, is particularly free from any impurities that go into solution, but the fact that both chalcocite and carbonate are present makes it a rather difficult problem. The particular ore deposit upon which we worked was that of the New Keystone, but the same class of ore existed in the Miami, and also on the Inspiration. The problem therefore presented to us was either a straight leaching one, which would have to be preceded by complete dead roasting, or else it would be a mixed process, water concentration followed by leaching, or possibly leaching followed by water concentration; and then finally, if flotation came in, we might have to make three bites of the cherry—that is, use water concentration, flotation, and leaching. You can readily see there would be a great many combinations; and whether it would be desirable to leach first, then water concentrate, and then float, or whether it might be desirable to make the sequence different, we do not know. This Keystone ore averages about $2\frac{1}{4}$ per cent. copper, of which anywhere from 30 to 40 per cent. is oxidized. After making a number of concentration tests we decided upon attempting a dead roast; in other words, converting all the copper into a soluble form and leaching it.

It is not possible for me to give you the details of the numerous different things we tried. I may say, however, that for the purpose of precipitation, we tried the electrolytic method. The problem with respect to these mixed ores is by no means solved and we are still conduct-

ing experiments along these lines, and I think Mr. Canby, the gentleman in charge of the experiments, can give you in a general way some idea of what we have accomplished and what is still left undone.

R. C. CANBY, Wallingford, Conn.—The only thing I can think of which has not been mentioned is the circulation of the electrolyte. In our electrolytic work I designed the cells so that the solution, instead of flowing lengthwise against the flat surfaces of the anodes and cathodes, flowed crosswise of the cell. In that way, in the use of sulphurous acid gas, I had hoped, by having a rapid circulation, to use a much smaller quantity of both copper and the depolarizing element in the electrolyte, with a correspondingly stronger current, and also to be able to precipitate the copper without heating the electrolyte. I was able to use a current density as high as 10 or 12 amperes per square foot with the temperature of the electrolyte about 18° to 20° C., and get a very satisfactory coherent deposit. I think that was the only novelty in the electrolytic work.

The other things were done very much along the lines which have been discussed here this evening. We think, however, it would be very much better if all of the oxides could be treated in the present plant, and we are now working to that end, so as not to have a separate installation for the semi-oxidized ores. It is along that line that experiments are now being carried on, so as to treat the oxidized material, which is mostly in the flotation tailings. I think there were perhaps five or six different methods of manipulation tried, such as leaching resulting slimes and leaching before crushing for concentration, etc., but all of these combinations contained really nothing of any special interest, as they are thoroughly familiar to everybody.

J. PARKE CHANNING.—How about roasting with the oil furnace—would that be of interest?

R. C. CANBY.—That might be of interest. The problem was to use our 14° Baumé fuel oil, to be burned at a temperature of 2,000°, and yet keep our ore bed at not to exceed 800° to 1,000° C. I accomplished this by using a furnace similar to that used for the Huntington and Heberlein process at El Paso, and at three other American smelting plants, the hearth revolving past the flame, so that I was able to roast the ore without at any time bringing the temperature of the roasting ore above the desired 1,000° C., whereas the fuel was being burned at about 2,000°, or a little over. The furnace was entirely of reinforced concrete construction and stood well.

RICHARD LAMB, New York, N. Y.—During 1907 I designed and erected a leaching and electrolytic copper extraction plant at High Hill, Va., on the Virginia Copper Co.'s property, to treat the bornite and chalcocite ores found there.

The average ore analyzed about as follows:

	Per Cent.
Copper.....	3.00
Sulphur.....	1.50
Silica.....	89.00
Lime.....	0.75
Iron.....	1.50
Magnesia.....	3.00
Alumina.....	0.35

Owing to the large percentage of silica this ore is difficult to concentrate economically. The large bulk of silica crowds off the values on any form of concentrating apparatus, and at least 50 per cent. of the values is lost if machines are worked at a speed necessary to make them efficient. There is a large body of copper ore in the Virginia copper belt, but many failures have been made in the attempt to concentrate it. However, the ore is ideal for leaching. The magnesia and alumina are mostly in silicate form, and do not go into the acid solution, and the large percentage of silica is in its favor when leaching.

I built the first experimental plant for treating this ore in Hoboken, and first tested leaching the ore in open tanks. In crushing this ore to say 10 mesh at least 25 per cent. goes into slime. This adds to the difficulty of leaching if the usual leaching tanks are used. I therefore substituted a chlorination barrel for the leaching tank in order that the crushed ore might be stirred while leaching. Stirring arms in open leaching tanks when using sulphuric acid disintegrate quickly. The lead-lined barrel used stirs the ore without injury from the acid. As the ore is a sulphide it has to be roasted. In the Hoboken test plant I used a small Brückner roaster. In the large test plant at the Virginia Copper Co.'s mine I used an Edwards roaster with a capacity of 30 tons of ore per 24 hr. I was able with this roaster to control the heat. While we had only 1.5 per cent. of iron in the ore, to avoid its polluting the electrolyte I sought to change it from an oxide to a sesquioxide of iron, which is insoluble in dilute sulphuric acid, by keeping the heat while roasting to about 900° F. Dr. James Douglas told me that he had succeeded in changing the iron to a sesquioxide of iron in a copper ore he was leaching, having over 25 per cent. of iron in same, by roasting with the proper degree of heat. I found that if the ore is crushed only to a size that exposes the sulphide on the surface, the roast will be as good as it would be if the ore was ground exceedingly fine. There is some oxide and carbonate of copper ore in the form of cuprite, malachite, melaconite, and green carbonate produced, but these are not separated when coming out of the mine, so they were sent to the roaster, although the roasting was superfluous, as sulphurous acid will take the copper of those oxide and carbonate ores into solution without their being roasted. The ideal

roast would be to change the sulphide ores into sulphate, but since this condition would require careful manipulating I sought to make the sulphide ore oxide and sulphate, but not to make a sweet roast, only seeking to leave as little of the ore in a sulphide condition as possible. If the ore could all be roasted to a sulphate condition the sulphur of the ore would go into the solution while leaching and thus add sulphur to the acid supply.

The ore, ground to 10 mesh and roasted, was put into a lead-lined barrel. A solution of sulphurous acid was made by burning sulphur in an inclosed oven with a supply of air sufficient in volume to supply the oxygen to produce sulphurous gas, and under a pressure just sufficient to overcome the hydraulic head due to the height of the water from the submerged outlet of the sulphurous gas delivery pipe to the surface of the water in the tank. The water is acidulated to about 10° Baumé by absorbing the sulphurous gas.

This acid is pumped into the leaching barrel with the roasted ore, and the barrel is revolved until the copper has gone into solution. This takes about 3 hr. Compressed air is then passed into the barrel and the solvent is forced from the barrel through the filter in same. This filter is the same as is used in the barrels used in chlorination work. This solution entered a 20,000-gal. cypress stock tank, painted with Mogul acid-proof paint. The elevation of the tank was such that gravity would produce a flow of the solution or electrolyte from the stock tank through 16 lead-lined cypress vats, placed at elevations with reference to each other so as to continue the flow of the electrolyte through the 16 vats. An acid-proof pipe line was also provided, situated below the vats, so that any vat could be discharged or shunted. Each tank was provided with $\frac{1}{8}$ -in. thick lead anodes and cathodes, with exposed surface 25 by 36 in., in all 1,030, the cathodes providing over 6,000 sq. ft. of depositing surface. An overhead crane was provided to carry the anodes and cathodes to their respective places.

The electric current was supplied by two Holtzer-Cabot 30-volt, 1,200-ampere generators, with a 120-volt, 1.5-kw. exciter.

By the use of sulphurous acid the anodes were prevented from peroxidizing. The sulphurous acid of this electrolyte is changed to sulphuric acid by the electrolysis and becomes a stable acid. After passing through the vats any loss of acid from being neutralized by the alkali of the ore is replaced by sulphurous acid, as before described, and the electrolyte is used over and over again to leach more ore. When the electrolyte becomes polluted it is run into a larger vat having an extra number of electrodes so that the current density is very slight and the copper is practically depleted. The electrolyte from this hospital tank can be run to waste or further treated for its sulphur contents.

The voltage supplied to each tank was 1.8, and a current density of

3 amperes per square foot was used. The electrolyte was permitted to run from $\frac{1}{4}$ to $\frac{1}{2}$ per cent. in copper, when used over for further leaching.

The output was 30.6 lb. of pure electrolytic copper per horsepower-day of 24 hr. With water power at \$12 per horsepower per year this would represent a cost of 0.12c. per pound for electrolytic treatment.

With steam power costing \$48 per horsepower per year the cost for electrolytic treatment would be about $\frac{1}{2}$ c. per pound of copper produced. Producing copper in the Virginia copper belt, from underground to finished product, would cost about 6.9c. per pound working upon a scale of say 5 tons of copper per day. In my belief, using the large available water power nearby, and making 5,000 tons per year, the cost of copper will not exceed 6c. per pound. The capacity of the trial plant described was 2,800 lb. per 24 hr. In building this plant I was employed by one man, who alone, on the advice of his brother, an experienced metallurgist, undertook to lease the Virginia Copper Co.'s mine and put up the experimental plant. The plant was run only a few days, the owner having lost his money in the fall of 1907, and the plant was thrown into the courts, with disastrous results to the owner. Enough was done to demonstrate that with the above described combination of well-known and tried-out apparatus, using the cheapest of solvents, sulphurous acid, there is no reason why a suitable copper ore should not be treated on a large scale so as to produce electrolytic copper at a lower cost than by the old dry-process methods.

DISCUSSION*

FREDERICK LAIST, Anaconda, Mont.—I am going to tell you of some of the results we have been getting in Anaconda during the past two or three months in a large plant which was finished in May. We commenced work in Anaconda a year and a half ago on the construction of a leaching plant which was to have a capacity of 2,000 tons per day. The plant was to treat tailings from the concentrator, containing about 12 lb. of copper to the ton. During the past year we changed our plans and decided to use the flotation process on the tailings from the concentrator, which contained no oxidized copper, and apply the leaching process only to tailings which had accumulated in a large dump below the works. This dump contains about 15,000,000 tons of tailings, and since some of the copper in it is oxidized, we felt that the application of the flotation process would result in the loss of too much of the oxidized copper, particularly as time went on. In starting up we had a number of difficulties to overcome, as is usually the case. They were not of a serious character. The results obtained, so far, have been decidedly encouraging.

During July the plant treated a total of 50,000 tons of tailings, or an

* Presented at the San Francisco Meeting, September, 1916.

average of 1,610 tons per day. The extraction was about 83 per cent., which, on material of the grade treated, meant a tailing assaying under 0.1 per cent. copper. The acid consumption was about 65 lb. per ton, and the fuel percentage in roasting was about $4\frac{1}{4}$. We expect to cut the fuel percentage in the next month or two to about 3 per cent., the high percentage during July being due to starting up and shutting down furnaces.

We find that iron builds up in the solution to a marked degree. This is not a serious matter with us, because we have cheap sulphuric acid and can reject solution whenever it gets too foul.

The extraction of silver has been remarkably good in the large plant. It has averaged better than 75 per cent. It seems that so long as we maintain salt in the solution to a percentage of 10 per cent., the extraction is good. We allowed the salt to fall off in order to find how low a percentage we could use. At about 7 per cent. the silver extraction fell to 65 per cent. This was to be expected, because silver chloride is soluble in strong brine, but not in weak brine.

T. T. READ, Engels, Cal.—Concerning the use of SO_2 as a depolarizer, it has occurred to me, though I have not been able to try it as yet, that a hollow porous carbon anode could be used, and the SO_2 could be made to pass outward through its pores, meeting the liberated oxygen on its outside and concentrating the action at the point where it is needed, instead of attempting to disseminate the SO_2 throughout the electrolytic bath, with the difficulties that produces. By using a sufficient pressure to force the SO_2 through the pores the difference in hydrostatic pressure between the top and bottom of the bath should not exhibit itself to a noticeable degree. If waste gases were used for this purpose they would be forced through in sufficient volume to make a strong upward current along the face of the anode, thus inducing a downward one along the cathode, so furnishing the required agitation of the electrolyte. As I say, I have not yet tried this, because I have not been able to procure any hollow porous carbon that seemed well adapted to what is required.

E. H. HAMILTON, West Norfolk, Va.—We have extracted and recovered over 80 per cent. of the copper from chemical works cinder without adding any new acid.

The acid builds up in the solution and can be kept normal by adding cinder high in zinc.

The residue is sintered and used as an iron ore.

Roasting and Leaching Concentrator Slimes Tailings

BY LAWRENCE ADDICKS, DOUGLAS, ARIZ.

(San Francisco Meeting, September, 1915)

THROUGH the courtesy of Dr. James Douglas I am permitted to give a summary of some of the results obtained in leaching slimes tailings in a series of experiments conducted under my direction at Douglas, Ariz., during the past year.

While the test run for which the data are given deals with a particular case, much of the work is general in its application and may therefore be of interest.

This work was done upon the ores of the Burro Mountain Copper Co. A description of the geology of the district will be found in the paper by R. E. Somers published in this volume, p. 604.

The mineral is pyrite and chalcocite very finely disseminated throughout the gangue and a characteristic of all the ores is the large amount of alumina present.

A representative analysis of the particular ore under consideration would be about as follows:

Cu, per cent.....	2.35	S, per cent.....	3.0
Ag, ounces per ton.....	0.05	CaO, per cent.....	0.5
Au.....	None	K ₂ O, per cent.....	4.0
Fe, per cent.....	1.50	SiO ₂ , per cent.....	70.0
Al ₂ O ₃ , per cent.....	13.00		

The leaching of the ore direct was early dismissed from consideration for the following reasons:

1. The ore would have to be quite finely crushed to liberate the mineral for leaching; concentrating after crushing would cost but a few cents a ton; and a higher total recovery would result from leaching tailings rather than ore.

2. Rough concentration would yield a suitable material for roasting to make sulphur dioxide or sulphuric acid in connection with the leaching scheme.

3. Such a program put the question of cost vs. recovery on a basis where flotation and discarding of tailings were directly comparable.

Concentrating experiments already available indicated that the tail-



FIG. 1.—GENERAL VIEW OF LEACHING PLANT.

ings would consist of about 50 per cent. of sands running about 0.3 per cent. in copper, and therefore too low to justify re-treatment, and 50 per cent. of slimes running perhaps 1 per cent. in copper, and it was on this latter material that the large-scale work was finally done.

At first flotation gave very unsatisfactory results, due primarily to the readiness with which the fine mineral particles became coated with a film of oxide upon exposure while concentrating, but later developments in this field overcame these difficulties and it became apparent just about the time the leaching plant was being started that the concentrator recovery was going to be so high that re-treatment of the tailings would not be justified. Nevertheless, for the sake of the data it was decided to carry through a representative run, the results of which are discussed below.

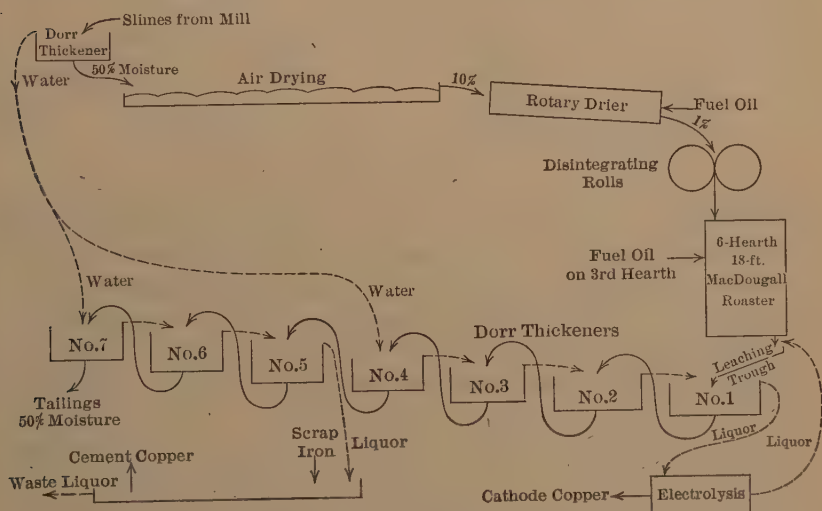


FIG. 2.—FLOW SHEET OF LEACHING PLANT.

Fig. 1 is a view of the leaching plant, and the general scheme is shown in the flow sheet, Fig. 2. As the development of the electrolytic precipitation is really a subject apart from the leaching of the slimes, the results from this part of the work have been reserved for a separate paper. A part of the copper is shown recovered as cement in order to give a means of holding the iron and alumina in the liquor to be electrolyzed down to the desired point. A high ratio of washing in the remaining thickeners delivering to the cementation launder; in the remaining thickeners the ratio is governed by the evaporation loss, as there is no other outlet for liquor.

The experiments will be taken up in detail under the headings of: (1) drying, (2) roasting, (3) leaching, and (4) washing.

Drying

As the slimes would be delivered from the concentrator in suspension as a thin pulp it became necessary to consider carefully the question of drying.

Thickening experiments indicated that there was a critical degree of moisture of about 45 per cent., below which pumping the settled slimes resulted in further classification of the fine particles instead of delivering

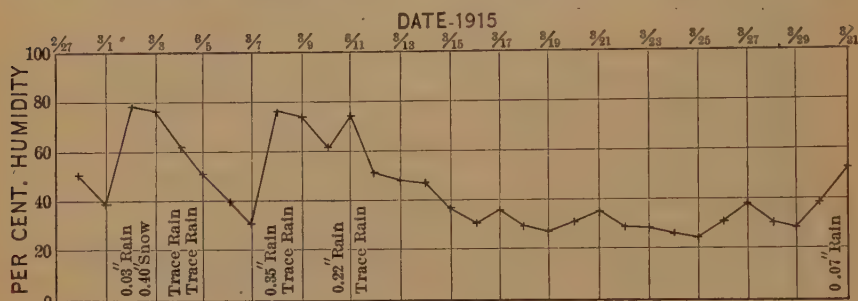


FIG. 3.—MOISTURE IN AIR.

average pulp, so that thickening in a standard Dorr unit to 50 per cent of moisture was established as the first step.

As fuel is very expensive at the location in question and no waste heat can be counted on except that from the roaster gases, experiments were made with drying by exposure to the desert sun and winds. The heavy pulp could be poured into a shallow basin and the more or less dried slime reclaimed by some mechanical method for much less expense than

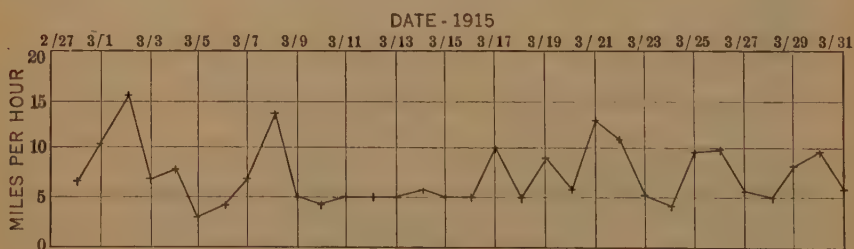


FIG. 4.—WIND VELOCITY.

an equivalent drying using artificial heat would cost. It was found that the pulp settled to about 30 per cent. of moisture very readily when allowed to drain and that reasonably dry material could be obtained by a month's standing.

Figs. 3, 4, 5, and 6 give the measurements taken during the trial on three test beds, 10 ft. square and respectively 2, 6, and 10 in. deep, which were exposed to the atmosphere for about a month.

The climatic conditions are shown in Figs. 3, 4, and 6 and it will be noted that periods of both cold and wet weather were encountered during the first part of the run, much more favorable conditions following in the last two weeks. A bed only 2 in. deep would cost a good deal to reclaim, but the rate of drying, as shown in Fig. 5, was thought to indicate that a

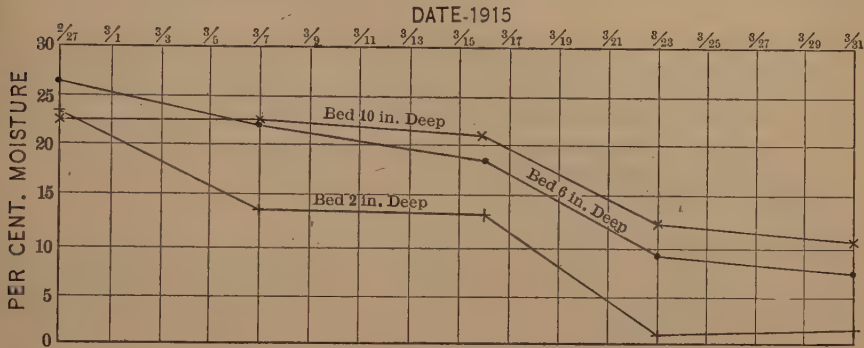


FIG. 5.—MOISTURE IN SLIMES.

reasonably thick bed could be brought down to 10 per cent. of moisture without an excessive period of standing, and it seems probable that this will be the cheapest way to obtain this preliminary drying.

Up to this point the experiments were conducted on relatively small quantities of material, but starting with the following step—drying in the rotary kiln—large tonnages were handled. Instead of using atmospheric

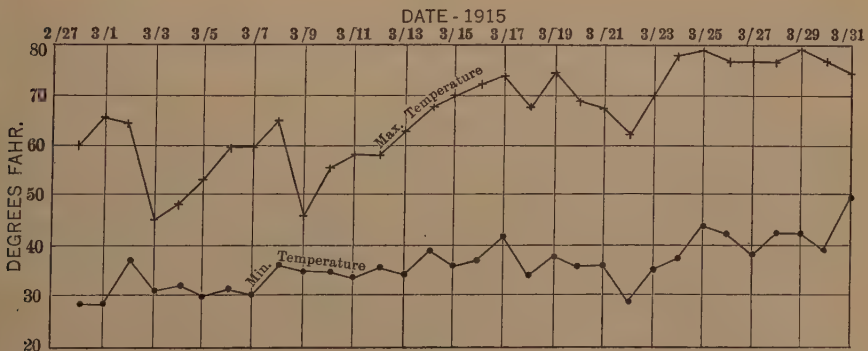


FIG. 6.—TEMPERATURE OF AIR.

drying before feeding to the kiln, steam coils were buried in the cars of mushy slimes shipped to the experimental plant, bringing the moisture down to about 25 per cent., when it was shoveled directly to the drier.

This drier was one of the old Douglas roasters, 7 ft. in diameter and 31 ft. long. It was used as an ordinary unlined rotary kiln, with channel-

iron spilling blades throughout the first two-thirds of its length and cast-iron balls 3 in. in diameter feeding against a delivery screen made of sheet-iron plate with $1\frac{1}{2}$ -in. round holes. The idea was to dry and pulverize any caked nodules in the one apparatus. The outside was lagged with $1\frac{1}{2}$ -in. magnesia block and heat was supplied by three oil burners at the delivery end. Two typical runs gave the following results:

Days run.....	23.0	12.0
Total tons dried.....	710.0	388.0
Tons per day.....	30.9	32.3
Per cent. moisture entering.....	26.6	10.0
Per cent. moisture leaving.....	1.1	2.0
Gallons fuel oil per day.....	404.0	154.0
Gallons fuel oil per ton dried.....	13.1	4.8

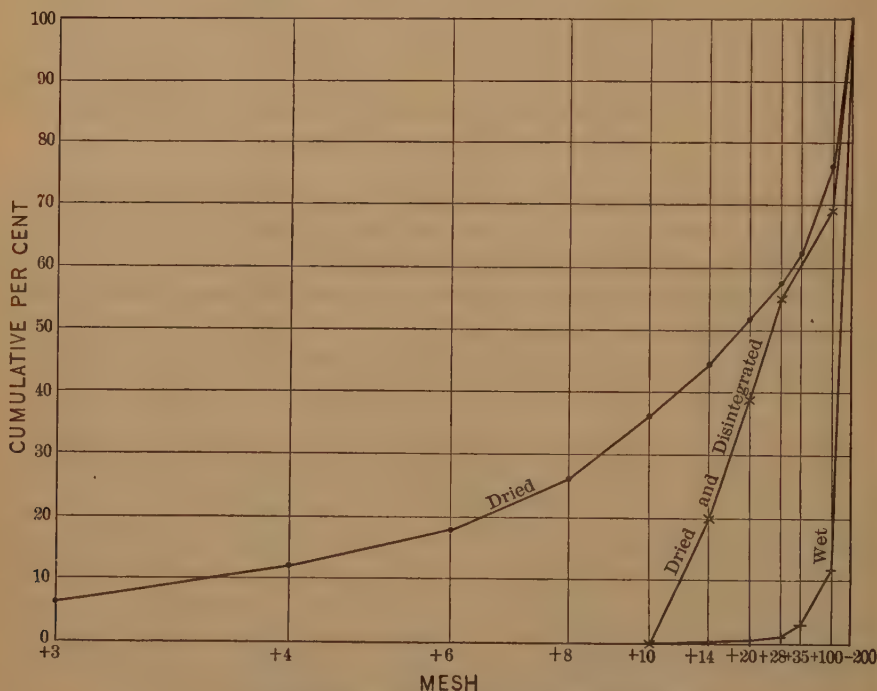


FIG. 7.—CUMULATIVE SCREEN ANALYSES OF SLIMES.

In general, it was concluded that about $\frac{1}{2}$ gal. of fuel oil must be provided for each unit of moisture in the ore to be dried, so that 5 gal. would have to be charged against drying a 10 per cent. product from the atmospheric drying beds, although this amount might be lowered in a properly designed drier. The capacity of the drier was too low for the roaster used and the pulverizing action of the iron balls inadequate, so the dried material was passed through a pair of disintegrating rolls, returning any

oversize through a 10-mesh screen, and dried and disintegrated material was stored ahead of the roaster.

Fig. 7 shows cumulative screen analyses of the original wet slimes, the drier product, and the disintegrated material, made on the usual Tyler direct diagram. It must be remembered that the lumps formed in the kiln could be readily crushed between the thumb and forefinger, being merely caked clay. The danger was that these lumps would be burned hard in the subsequent roasting.

The arrangement of the drier with skip feeding to roaster, and the row of Dorr thickeners, can be readily seen in Fig. 1.

Roasting

The roasting was carried out in a standard six-hearth, 18-ft. MacDougall furnace with water-cooled arms, fuel oil being introduced on the third hearth. No change was made in the furnace for this work except to

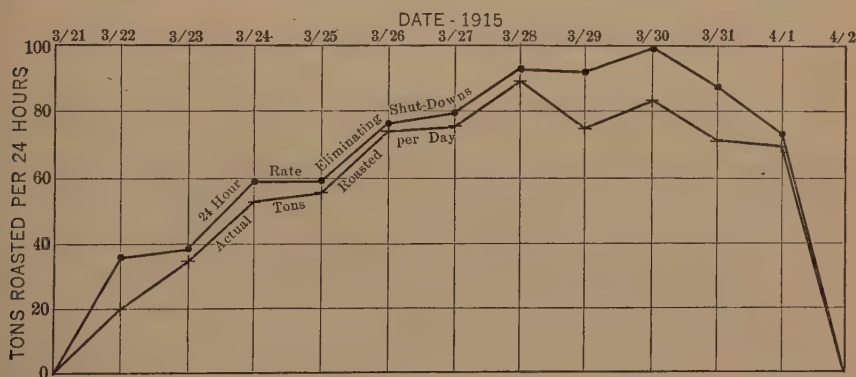


FIG. 8.—TONS ORE TO ROASTER.

replace parts of the upper brickwork with firebrick. Fig. 8 shows the tonnage fed to the furnace daily during the particular run with which the remainder of this paper chiefly deals. When the arms began to be buried the speed of rotation was increased from one revolution in 69 sec. to one in 60 sec. No trouble was experienced in the operation of the furnace.

Fig. 9 shows how the fuel-oil consumption was gradually cut down as the size of charge was increased. It is evident that 100 tons a day can be roasted with about 3.5 gal. of fuel oil per ton. The fuel oil burned was regulated so as to keep the maximum temperature on any hearth at about 950° F. Previous small-scale experiments had indicated that the proper roasting range lay between 900° and 1,100° F. Fig. 10 shows a log of the daily temperatures on the second, third, fourth, and fifth hearths. It will be noted that the heat zone descended in the furnace as the tonnage increased. This is shown diagrammatically in Fig. 11. At the same time

the efficiency of the roast increased, as shown by Fig. 12. This is apparently due to the fact that at the higher rates roasting temperatures were obtained on the fourth and fifth hearths, which were free from the more or less reducing atmosphere of the third and higher hearths due to the fuel oil.

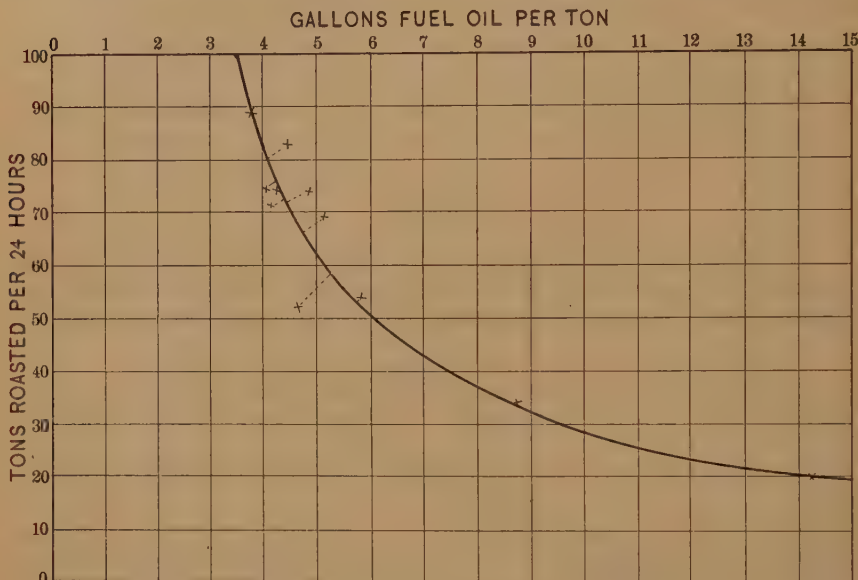


FIG. 9.—ROASTER TONNAGE VS. FUEL-OIL CONSUMPTION.

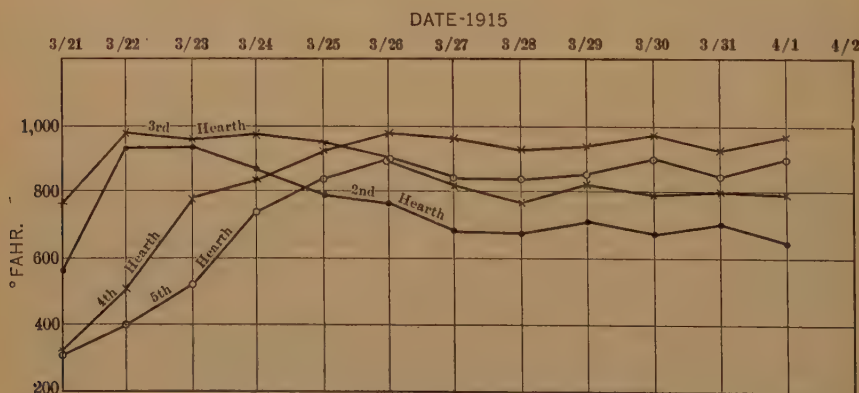


FIG. 10.—ROASTER HEARTH TEMPERATURES.

Leaching

The hot calcines from the roaster were delivered through a spout casting made of Duriron into a triangular trough about 30 ft. long sloping $1\frac{1}{4}$ in. to the foot, which delivered directly into the first of the series of

Dorr thickeners used for washing. This trough served three purposes: (1) that of leaching by agitation; (2) that of mechanically conveying the calcines from the roaster to the washing apparatus; and (3) that of absorbing the sensible heat of the calcines, thereby furnishing a warm liquor

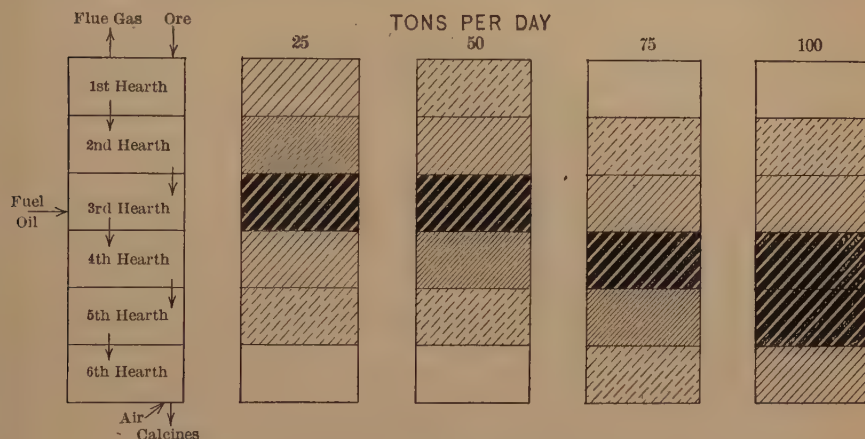


FIG. 11.—ROASTER HEARTH TEMPERATURES.

for electrolysis. A similar arrangement is described as doing excellent work in another field by Hofmann in his *Hydrometallurgy of Silver*. A deep triangular trough was first constructed of common lumber. This was then lined with sheet lead to make it liquor proof. Inside the lead

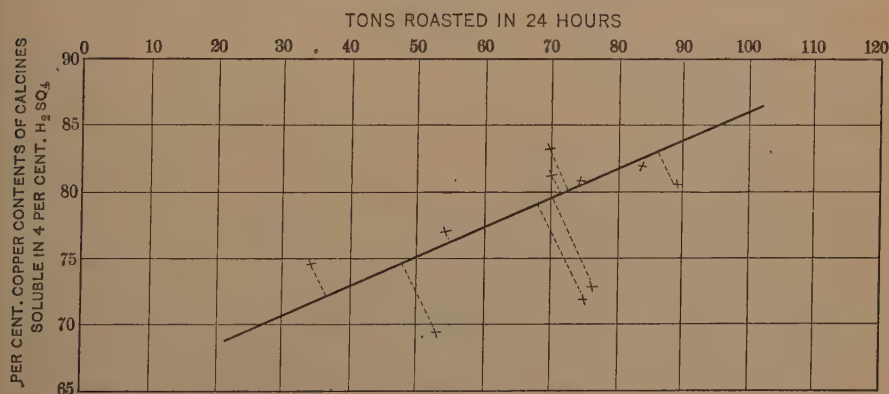


FIG. 12.—VARIATION OF ROASTER EFFICIENCY WITH TONNAGE.

lining, boards were loosely laid to prevent mechanical erosion of the lead by the calcines. A flat wooden cover was laid on top of the launder to keep down the steam and dust. This latter was not excessive, and would have been very slight had a more even delivery of calcines been made at

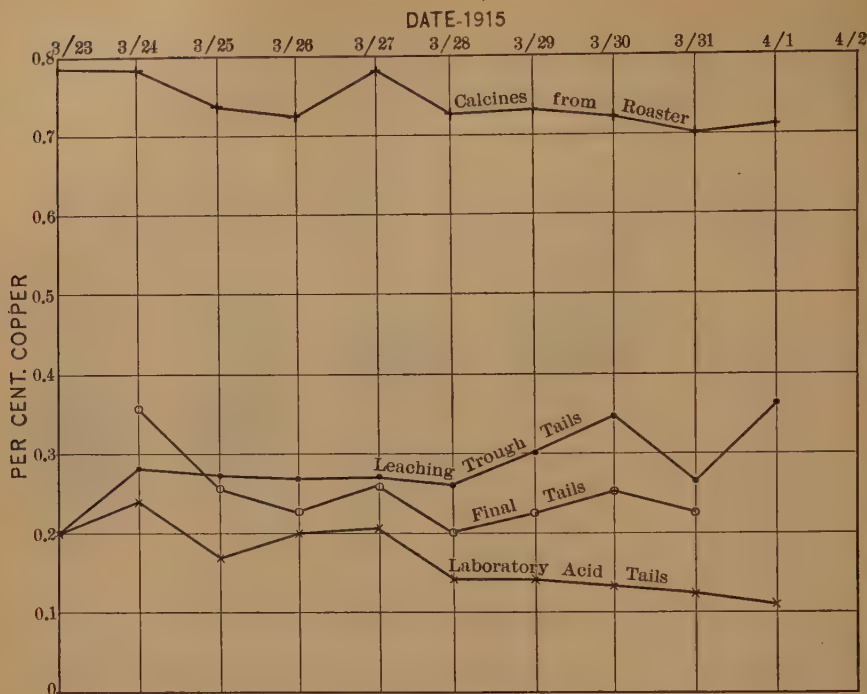


FIG. 13.—COPPER CONTENTS OF TAILINGS.

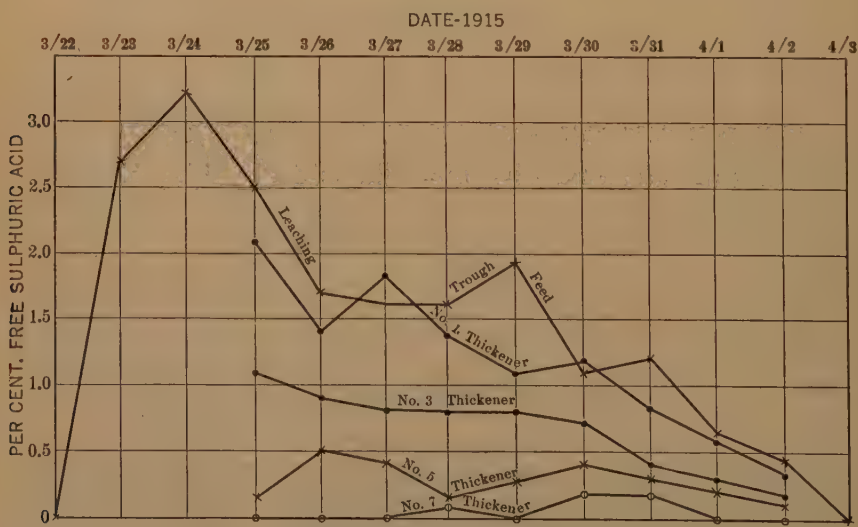


FIG. 14.—ACID IN LIQUORS.

the head of the sluice. The furnace arms alternated with light and heavy charges through the bottom drop hole and these were spread out over a reasonable time interval by placing an adjustable gate in the delivery spout, something on the plan of the classic hour glass. This worked fairly well, but it would have been better to have arranged some extra arms on the bottom hearth.

The copper content of the tailings at various points in their travel through the plant is shown in Fig. 13. The free sulphuric acid content of the liquor is shown in Fig. 14 and the effect of this strength of acid upon total extraction in Fig. 15.

The slimes as received assayed 0.84 per cent. of copper. Fig. 13 shows that the calcines averaged about 0.74 per cent. of copper, the grade falling as the furnace was gradually speeded up. This difference in assay was carefully investigated in the laboratory and it seems very improbable

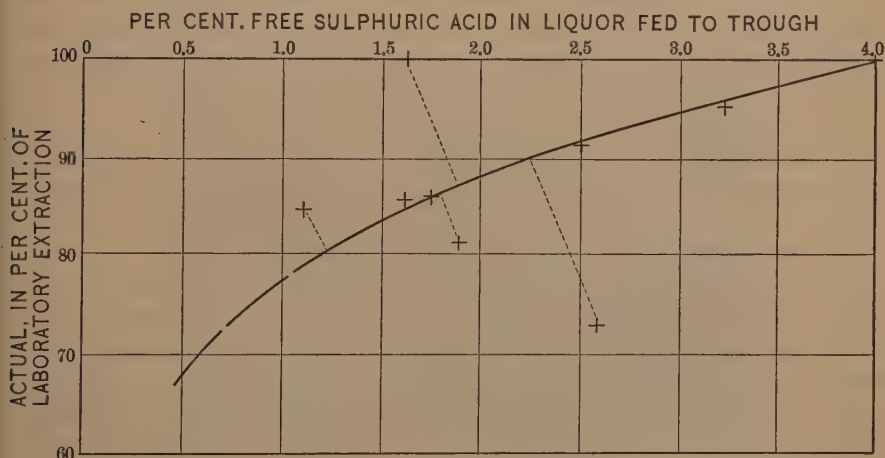


FIG. 15.—FREE ACID IN LIQUOR VS. EFFICIENCY OF EXTRACTION.

that it was due to the formation of insoluble copper compounds during the roast, as three acids were used in the assay and various methods yielded but the merest traces of copper from the assay residue. The most probable explanation lies in the formation of particles of anhydrous copper sulphate during the roast, which blew over with the flue dust. No measurements were made of the flue dust, as the furnace discharged into a common flue with other roasters treating concentrates.

The tails, as determined in the laboratory by heating samples of the calcines in 4 per cent. sulphuric acid, showed 0.2 per cent. of copper, gradually dropping to 0.1 per cent. as the furnace efficiency increased with the load. The actual final tails discharged from the last thickener ran about 0.1 per cent. higher than the laboratory-leached calcines and laboratory test showed this 0.1 per cent. to be present as soluble copper. This same

margin has been noted by Croasdale in his paper presented last year on his experiments at Ajo, and he there attributes it to the impossibility in practice of washing ore free from soluble salts. Fig. 15, however, seems to indicate that the difference in this case is largely due to the stronger acid liquor used in the laboratory tests.

Fig. 14 shows that the run was started with over 3 per cent. of free sulphuric acid at the leaching trough and that this strength was allowed to exhaust itself gradually upon the entering ore as the run proceeded, the difference between final actual tails and laboratory tails increasing as the acid dropped. This same ratio is the basis of Fig. 14. It seems probable, therefore, that the strength of acid used will have to be considered in its bearing upon efficiency of extraction.

The leaching trough discharge ran but a few hundredths of a per cent. higher in copper than the final tails after seven washings, showing that over 90 per cent. of the leaching was done in the few seconds taken to travel down the trough. This high efficiency is due to the violent agitation when the hot calcines meet the relatively cool liquor, and to the fact that each particle of solid meets a fresh entering drop of liquor at the head of the trough, obtaining the full efficiency of the acid strength. An actual over-all extraction in practice somewhere in the seventies seems assured from these experiments.

Washing

The separation of the dissolved values from the exhausted slimes was one of the main problems presented. Any method of percolation was out of the question owing to the extremely fine state of subdivision of the material. The acid liquors employed similarly ruled out many of the mechanical filters. It was finally decided to try out continuous counter-current decantation with Dorr thickeners of the acid-proof type of construction, some experience with which had been gained at the installation at the Butte-Duluth Mining Co.'s plant. Two 30 by 10 ft. and five 24 by 10 ft. tanks with suitable mechanisms and wood-stave pipes and fittings for 4-in. air lifts were therefore ordered and connected after erection in accordance with the flow sheet shown in Fig. 2.

The acid proofing of these mechanisms consisted of the substitution of wood for iron in the under-liquor parts. In turn, this called for some changes in design, two arms being used in place of four and the inclination of the swept floor being lessened. Both these changes increased the digging load and the arms tended to ride rather than dig, extreme cases overstraining the unit. In spite of these difficulties the thickeners proved a practical apparatus. I believe, however, that the original metal construction properly protected by lead and equipped with Duriron rakes would be an improvement. An idea of the wooden construction can be obtained from Fig. 16.

The main difficulty was with the air lifts and I am satisfied that these are not practical for the particular case. It must be remembered that it was desired to make an underflow of 50 per cent. solids and that in the first thickeners many small lumps of slime had to be handled. When an air lift was gradually throttled until the desired thickness was obtained it no longer acted as a pump but more like a cannon, and a small capacity was obtained, finally resulting in one of two undesirable climaxes. Either a few small lumps settled out in the horizontal pipe under the tank, due to the slow flow, thereby choking it, or the lift would run thin for a moment followed by a rush as the normal pumping action was restored, stirring up the settled slimes and requiring constant attention for the next half hour in order to get the desired thickness of underflow. Also, the scouring action on the wood elbow at the point where the air was introduced was excessive, pipes sometimes wearing through in 24 hr.

Simple spigot discharge into a launder with some mechanical means of elevating was not promising, again due to obstructions making the flow



FIG. 16.—THICKENER PADDLE.

of such a thick pulp very hard to regulate. Finally, at Mr. Dorr's suggestion two changes were made: (1) a diaphragm pump was tried, and (2) the remaining air lifts were arranged to return a part of the underflow to the same tank, thereby increasing the solids moved and keeping the rate of flow in the horizontal feed pipe above the critical speed at which lumps settled out.

The diaphragm pump was an entire success. In the first place, the delivery depends almost entirely upon the stroke and speed and is not influenced by a momentary change in thickness as in the case of the air lift. Then the suction has the entire pressure of the atmosphere at its command and this is exerted whenever an obstruction is encountered. In fact, an ore sack and a pair of gloves were pumped up, to the surprise of the operator. It is astonishing how many things find their way into a

system of this kind. A watch and fob and a brace and bit were among the articles added to our museum. Finally, the solids are moved steadily along and the scouring action is reduced to a degree which seems to make the use of wood pipe and fittings entirely practical. The pump was an ordinary 4-in. Gould contractor's pump with cast-iron body. The metal parts could readily be made of Duriron or other acid-resisting material.

During the run the temperature of the liquor in the first thickener gradually rose from 56° F. at the start until 100° F. was reached on the sixth day, at which point it was maintained by the sensible heat of the entering calcines. It was found possible to settle as high as 125 tons of solids in 24 hr. in one of the 30-ft. tanks. The overflow by actual test showed but 3 lb. of solids to the ton of liquor.

The liquors during the run accumulated sulphates corresponding to 10.0 lb. copper, 4.3 lb. iron, and 16.2 lb. alumina per ton of slimes leached. The actual acid consumed was 49.0 lb. The acid equivalent of the corresponding metallic oxides is 70.5 lb., showing the presence of considerable sulphates in the calcines. A laboratory test on the calcines, using 4 per cent. acid, showed a consumption of 68.9 lb. and a similar test on the product of a small-scale roast 69.1 lb. acid per ton. It appears, therefore, that less acid will be used in practice than in laboratory tests. Unfortunately, the run was not sufficiently prolonged to determine whether the proportion of impurities dissolved decreased as the liquors became more concentrated.

DISCUSSION

L. D. RICKETTS, New York, N. Y.—In the Southwest we have stored up millions of tons of tailings in which the copper exists partly as sulphide and partly in an oxidized condition. Mr. Addicks' first paper suggests a likely process for the treatment of such material. His second paper¹ deals with the recovery of copper from clear solutions of notable strength, and such solutions may come from the sulphating roast or direct from the leaching of completely oxidized copper ores of which vast deposits are known to exist.

What strikes me particularly about the latter paper is the keen perception shown and the clear manner in which the writer has attacked the fundamental principles of the process. He calls your attention to the fact that ferrous sulphate is necessary in the electrolyte and is an ideal depolarizer; that sulphurous acid must be used to reduce the ferric sulphate produced and thus prevent re-solution of copper at the cathode; that mechanical agitation in the cell, in order to bring the ferrous sulphate in contact with the anode and the copper in contact with the cathode, is essential to efficient work; that lead anodes are

¹ The Electrolysis of Copper Sulphate Liquors, Using Carbon Anodes, published by the American Electrochemical Society.

undesirable because they do not permit complete depolarization; and that carbon anodes are ideal for the purposes. When the ferric salts were kept low by the use of sulphurous acid there was no trouble during the life of his experiments from softening and deterioration.

E. H. HAMILTON, West Norfolk, Va.—Does sulphuric acid accumulate?

LAWRENCE ADDICKS, Douglas, Ariz.—That is a relative question. We had so much alumina, and the rate of neutralization was so high, it was more a question of getting enough acid than too much. You have raised a large question. In the process we were attempting we are bound to get whatever excess acid equivalent there is, in liquors contaminated with other things where it is of no use. If we have any acid left over, there is no market for it, consequently no revenue to be derived from it, which is the whole question.

FREDERICK LAIST, Anaconda, Mont.—Mr. Addicks spoke of the difficulty of getting a strong solution of SO_2 from a weak gas, such as would be encountered in practice. I believe Doctor Ricketts mentioned that point in connection with the paper he presented on Some Problems in Copper Leaching. I think it might be of interest to give a brief outline of some of the experiments we made in Anaconda in the course of the past year and a half, on the precipitation of copper from cupric chloride solutions, by means of SO_2 . While the plan decided on for our tailings treatment uses scrap iron as a precipitant, we wished to make sure that really would be the best method. Among other methods investigated was the old Hunt and Douglas method, which consists of adding to the copper sulphate solution sufficient common salt to chloridize the copper. The copper is then precipitated by SO_2 gas in the form of cuprous chloride, which is a white insoluble precipitate containing about 65 per cent. copper.

We carried on the process in two stages. First, we dissolved sufficient SO_2 gas in the solution. Second, we heated the resulting liquor to 90°C . which caused reduction of cupric to cuprous chloride. The latter separated out in cooling. Our solution contained about 1.5 per cent. copper and required about 0.7 per cent. SO_2 for precipitation. We found it impossible to dissolve over 0.2 per cent. at atmospheric pressure when a 10 per cent. gas was used. After we started using a closed tank for the work and forced the gas in under a pressure of about 15 lb. per square inch we obtained the desired saturation without trouble. There was no trouble about getting a 10 per cent. gas from the furnace although it was necessary to take the gas from the second floor in order to avoid admixture of air entering through the feed openings. With suitable apparatus 80 per cent. of the SO_2 , or better, could be absorbed.

The point to which I wish especially to call attention is the greatly increased strength of solution it is possible to get from a weak gas by the use of moderate pressures.

LAWRENCE ADDICKS.—I have never had any difficulty with the absorption towers, nor did we have any difficulty in getting as much gas absorbed as we did in the laboratory. We put in a stand pipe, 40 or 50 ft. high, and let the liquor run through that, obtaining absolute absorption of SO_2 . While there was no increase in efficiency, we obtained what we expected from the increased pressure.

CHARLES BUTTERS, Oakland, Cal.—Do carbon anodes soften when used as depolarizing anodes in a cyanide solution?

LAWRENCE ADDICKS.—With chloride electrolysis there has never been any difficulty. It has no oxidizing effect on the carbon. In the operation you mention, the efficiency is low since a great deal of oxygen is liberated, and carbon would certainly be attacked. You must take care of the product of depolarization.

L. D. RICKETTS.—In the Southwest we have porphyries that have been subjected to intense chemical action and contain a great deal of clay. In the old days, when we used to crush to $1\frac{1}{2}$ mm., fully 50 per cent. of the product would pass a 200-mesh screen. I think such ores would give a great deal of difficulty in leaching, even after a sulphating roast, if an attempt were made to gain a strong solution of sulphate of copper, but possibly we might make the process successful if we used a large amount of solution and produced a solution weak in copper from which the copper would be recovered with scrap iron and metallized calceine. Though such a method of recovery of copper from solution is comparatively expensive, the metal would be obtained very profitably because the material to be treated is tailings that are already mined and that are comparatively rich.

Referring to Mr. Addicks' second paper, and especially referring to the use of sulphurous acid to reduce the ferric iron produced by depolarization, at plants located at points like Douglas, where immense amounts of sulphur are going to waste as sulphurous acid gas, the question of efficiency does not enter into the problem. In certain districts in the Southwest, especially at Ajo and Globe, Ariz., we would have to consider the importation of pyrite or sulphur for the manufacture of the sulphurous acid gas. In such cases the question of complete absorption becomes very important, yet complete absorption of the gas as produced by burning sulphur or pyrite presents many difficulties. In our experiments we did not get very good absorption in a coke tower. The solution possibly followed channels. At all events we found if we could spread out the liquid in very thin sheets over large surfaces we obtained much greater

absorption. We also found that a neutral solution absorbs more gas than an acid solution. Where, however, complete absorption is important, it occurs to me that the pumping cost will be considerable and the cost of the construction of the necessary absorption towers will be very great.

Mr. Addicks' paper calls attention to the fact that if solutions containing ferric salt and sulphurous acid are heated to a point about 130°F., the reduction is almost instantaneous. I think the question of obtaining a pure gas is important. If by some cheap method 100 per cent. sulphurous acid gas could be obtained, the necessary amount could be pumped under a low pressure into the warm solutions and the reduction might be instantaneous, and the cost of both operation and construction might be very considerably lessened.

T. T. READ, Engels, Cal. (communication to the Secretary*).—Mr. Addicks' paper (Electrolysis of Copper Solutions with Carbon Anodes) is extremely suggestive and helpful, and I find myself in special accord with his method of attack as I have always believed that copper hydrometallurgy would eventually develop along the lines of cyanide practice, namely, fine grinding and the treatment of such material by the methods it requires. This seems even more true now that flotation is becoming general, for fine grinding is a prerequisite to that, and it seems quite likely that practice of the future will develop along the lines of a combined flotation-hydrometallurgical process. It is fairly easy to float sulphides; it seems foolish, therefore, to roast them at considerable cost, preparatory to an acid treatment that is no more satisfactory than flotation. Oxidized minerals yield easily to acid treatment; it seems at least doubtful whether they can ever be converted into sulphides and floated, thus putting them into concentrate that requires a subsequent treatment, and still show a profit in the operation. Of course there is the advantage of such a scheme that you are getting the desired mineral in a concentrated form, whereas hydrometallurgy is fundamentally unsound in that you take ore that contains only a few pounds of copper per ton and dissolve that copper out into three or more tons of solution, thus diluting it to one-third of its original dissemination. The first step in hydrometallurgy is not, therefore, concentration, but dilution, and is fundamentally bad. On the other hand, the problems of hydrometallurgy are largely mechanical and the advantages of handling the ore in the form of pulp are so great that it seems to more than counterbalance this drawback. On the whole it seems likely that the treatment of copper ore by sulphuric acid will develop along the same general lines as cyanidation, and we will do well to direct our attention where Mr. Addicks has, to the treatment of slime tailing.

It would be well, by the way, if we had a better name for this field of work. It is now commonly called copper leaching, but that is a misnomer, for the work Mr. Addicks has described does not involve leaching at all, and even where leaching is used it is not the most important of a series of operations that are necessary. Hydrometallurgy of copper is cumbersome and too general in its scope to define exactly the treatment of copper ore with sulphuric acid and the subsequent recovery of the copper from sulphate solution. It would be well if some one introduced a new word of precise meaning, just as cyanidation is perfectly definite. Possibly vitriolization would do, or perhaps some one will suggest a better word.

The Hydro-Electrolytic Treatment of Copper Ores*

BY ROBERT RHEA GOODRICH, TUCSON, ARIZ.

(San Francisco Meeting, September, 1915)

EXPERIMENTS ON A PORPHYRY COPPER ORE FROM BISBEE, ARIZ.

THIS research was done partly in the non-ferrous laboratory of the Department of Metallurgy of Columbia University, under the direction of Dr. Edward F. Kern, and completed elsewhere. Acknowledgment is due to Prof. Arthur L. Walker, Dr. Edward F. Kern and Dr. William Campbell of the Department of Metallurgy, for their kind advice and for the inspiration derived from their instruction.

This report is given under the following heads: Petrographic Description (Microscopic Study); Sampling and Preparing Ore for Treatment; Chemical Analysis; Treatment of Ore.

PETROGRAPHIC DESCRIPTION

Texture: Variable. Fine to medium mixed aggregate.

Original structure: Obscure.

Secondary structure: Fractured; healed.

Mineralogy

(Minerals are grouped for interpretation purposes and are arranged in each group in approximate order of abundance)

Primary, Essential Minerals	Primary, Accessory Minerals	Secondary, Alteration Products
Obscure Probably quartz, and feldspar now wholly altered	Zircon	Quartz Kaolin Sericitic
Introduced Substances or Mineralization	Tertiary Changes and Enrichment Effects on Ores	
Yellow sulphide (pyrite) Black sulphide (chalcocite) Quartz	Green malachite Limonite (a little from the yellow sulphide)	

* Submitted in partial fulfillment of the requirements for the Degree of Doctor of Philosophy, in the Faculty of Pure Science, Columbia University.

The original character of the rock has been much obscured by modification. The variable texture indicates that the original rock was much brecciated. Silicification seems to have been a prominent feature. Some of the quartz appears to be remnants of primary grains. It must have been primarily an acid intrusive, which, after having been fractured and brecciated, has become still more acid by silicification. There appear to be recorded several sets of movements. The special features of most importance seem to be as follows:

The Ground Mass

This shows variable texture, fragmental (brecciated). The whole is so much modified that the fragmental character is not plain. Most of

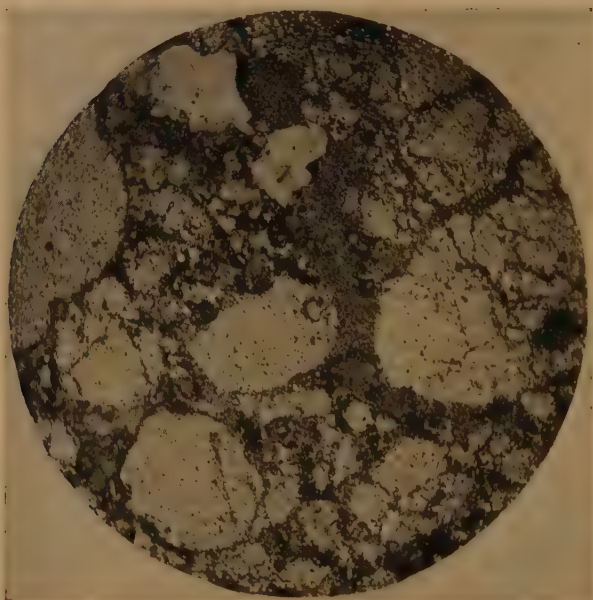


FIG. 1.—PHOTO-MICROGRAPH G₁.
Magnification 30 diameters.

the grains seem to be quartz. The only evidences of feldspar are those areas judged to be slightly kaolinized and areas of sericite. The fragmental character of the rock is somewhat further emphasized by the manner in which the metallics are distributed. These are distributed in such a way as to suggest an original fragmental (breccia), into the interstices of which were introduced the yellow and black sulphides. Certain streaked areas are much clearer than the rest of the ground mass and generally free from introduced metallics. These are filled with what is plainly

introduced quartz, and they seem to represent old fractures. It is difficult to say just what relation these bear to the mineralization periods; they may possibly represent the closing stages of silicification.

The Introduced Metallics

These consist of yellow pyrite and black chalcocite. The yellow sulphide seems to be a little too yellow for typical pyrite and not quite yellow enough for chalcopyrite. The yellow sulphide bears evidence of having been fractured and somewhat crushed. The fractures are healed with the black sulphide. The fractures extending across the grains of yellow sulphide end abruptly at their margins, and do not continue into

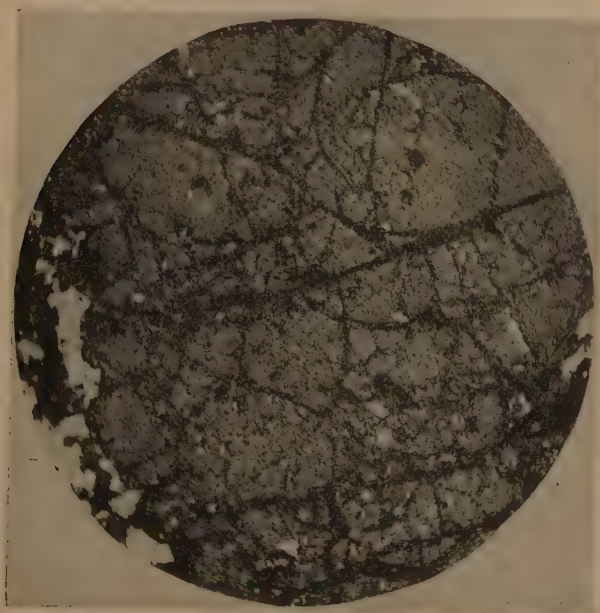


FIG. 2.—PHOTO-MICROGRAPH G₂.
Magnification 30 diameters.

the adjacent grains of ground mass. Under the microscope, there may be seen areas of yellow sulphide, veined and rimmed with black sulphide; the whole is rimmed with a narrow band of sericite and, in some cases, of quartz. It is evident that the history of this rock is very complicated.

It would seem, therefore, that two periods of mineralization are here represented: first, the decomposition of the yellow sulphide, followed by sufficient movement to cause fracturing and slight crushing, either during the closing stages of its deposition or immediately thereafter; second, the introduction of the black sulphide, which was deposited in the pre-

viously formed and fractured yellow sulphide and as an envelope surrounding the smaller grains of the yellow sulphide. Silicification very likely accompanied all of these changes. According to this interpretation, the rock is a silicified and mineralized brecciated, acid intrusive (quartz porphyry), in which two periods of mineralization are represented. There is no absolute proof in these slides as to the source of the secondary mineralization, *i.e.*, the chalcocite.¹

Photo-micrographs G₁, G₂, G₃XN, Figs. 1, 2, 3.

(Magnification 30 Diameters).

G₁—The large patches, light in shade, comprising the greater part of

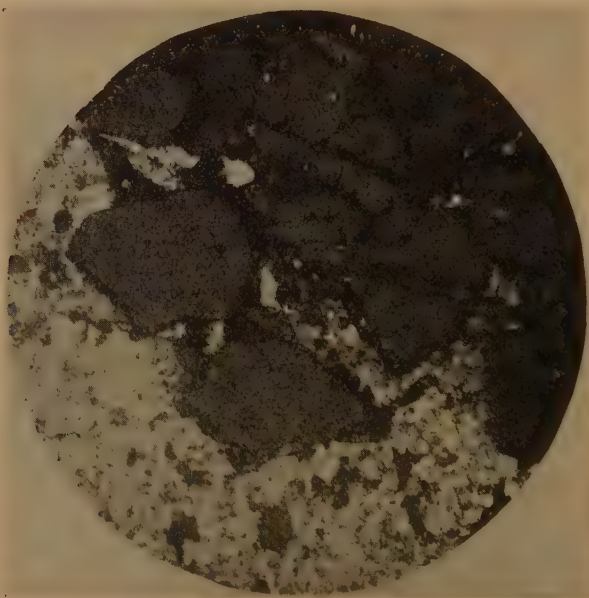


FIG. 3.—PHOTO-MICROGRAPH G₃XN.
Magnification 30 diameters.

the photo-micrographs, are yellow pyrite. There are two large patches, and several smaller ones, light in shade, but differing in appearance from those representing pyrite, in that they have a surface of homogeneous shade. These are quartz. The darker portions, occurring as spots and veinlets cutting the yellow pyrite, are of chalcocite.

G₂—In this photo-micrograph, the black masses and veinlets of chalcocite are very strongly marked. There is also some white non-metallic material.

G₃XN—This photo-micrograph was taken with crossed nicols of a

¹ This abstract was taken from the petrographic description by R. J. Colony, made under the direction of Dr. Charles P. Berkey, Columbia University.

slide containing more non-metallic material. The yellow pyrite here shows as large patches, quite dark, intersected by black veinlets of chalcocite. There is much white non-metallic material which, being doubly refracting, shows white as in the other photo-micrograph (taken with upper nicols out), while the pyrite, not being doubly refracting, shows darker.

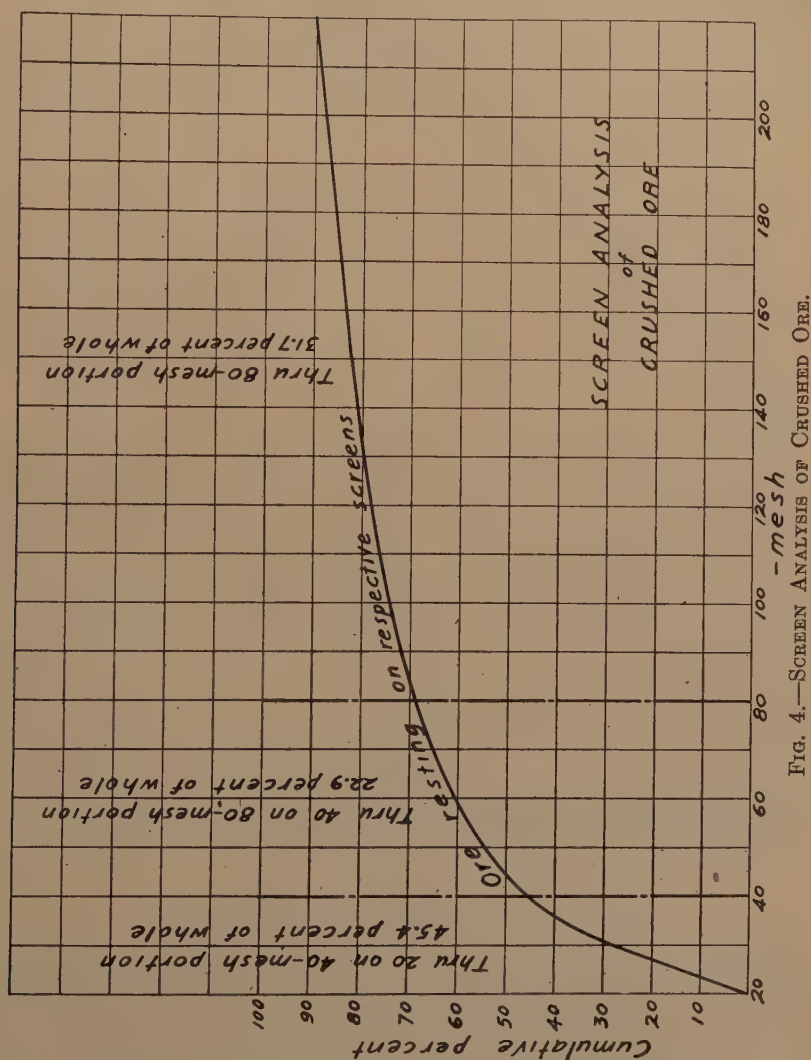


FIG. 4.—SCREEN ANALYSIS OF CRUSHED ORE.

SAMPLING AND PREPARING ORE FOR TREATMENT

The ore was received in lump form. Five hundred pounds were crushed to pass a 4-mesh screen, using the gyratory crusher followed by the cone-and-ring sample grinder. A sample for analysis was cut out

and ground through 100 mesh. One-quarter of the lot was then cut out for treatment. This was passed repeatedly through the sample grinder (which was set up so as to grind finer), until all passed a 20-mesh screen. This one-quarter was then thoroughly mixed by repeated coning, and then split by split shovel. One resulting one-eighth was retained for treatment and will be called "Whole through 20 mesh." The other resulting one-eighth was sized, producing the portions: Through 20 on 40 mesh, 45.4 per cent. of whole; through 40 on 80 mesh, 22.9 per cent. of whole; through 80 mesh, 31.7 per cent. of whole. (The screen analysis of the crushed ore is given in Fig. 4.)

Chemical Analysis

	Per Cent.	
SiO ₂ (insoluble).....	65.10	
Fe.....	10.90	
CaO.....	0.90	
Al ₂ O ₃	0.28	
Total Cu.....	6.04	{ 1.70 sol. in dil. HCl. 4.34 bal. of Cu.
S.....	12.70	

TREATMENT OF ORE

It has been seen that the ore is mainly a sulphide. Copper sulphide ores, which contain an excess of silica, have been heretofore usually treated by mechanical concentration followed by smelting. But this ore, like much of the disseminated orebodies of the Southwest, contains a portion of its valuable copper contents in an oxidized condition—malachite. Mechanical concentration on pure sulphides makes usually a saving of 66 per cent.; and when there is an oxidized component the saving is likely to be still lower. For this reason, it was decided to make a test on this ore by a leaching method. The method selected comprises:

1. Oxidizing roast;
2. Leaching with dilute sulphuric acid;
3. Electrolytic precipitation of the dissolved copper.

Oxidizing Roast

A series of four roasts was made in a gas muffle furnace (17½ by 11½ by 4 in.—inside measurement). In order that the relative roasting qualities of coarse and fine material, as well as the most suitable temperature, might be determined, 300 g. of through 20 on 40 mesh material and 300 g. of through 80 mesh material were roasted, in two 7-in. roasting dishes. The temperature of roasting was measured by a thermo-electric pyrometer. Samples to determine the progress of roasting were taken at ½-hr. intervals. These samples were ground through 100 mesh and then tested as follows: One gram of sample was boiled in a covered casserole for 20

min. with 150 cc. of water. The soluble copper thus obtained is reported in per cent. copper soluble in water. After filtering and washing, the same portion of sample was boiled in a covered casserole for 20 min. with 150 cc. of dilute hydrochloric acid (100 cc. of concentrated hydrochloric

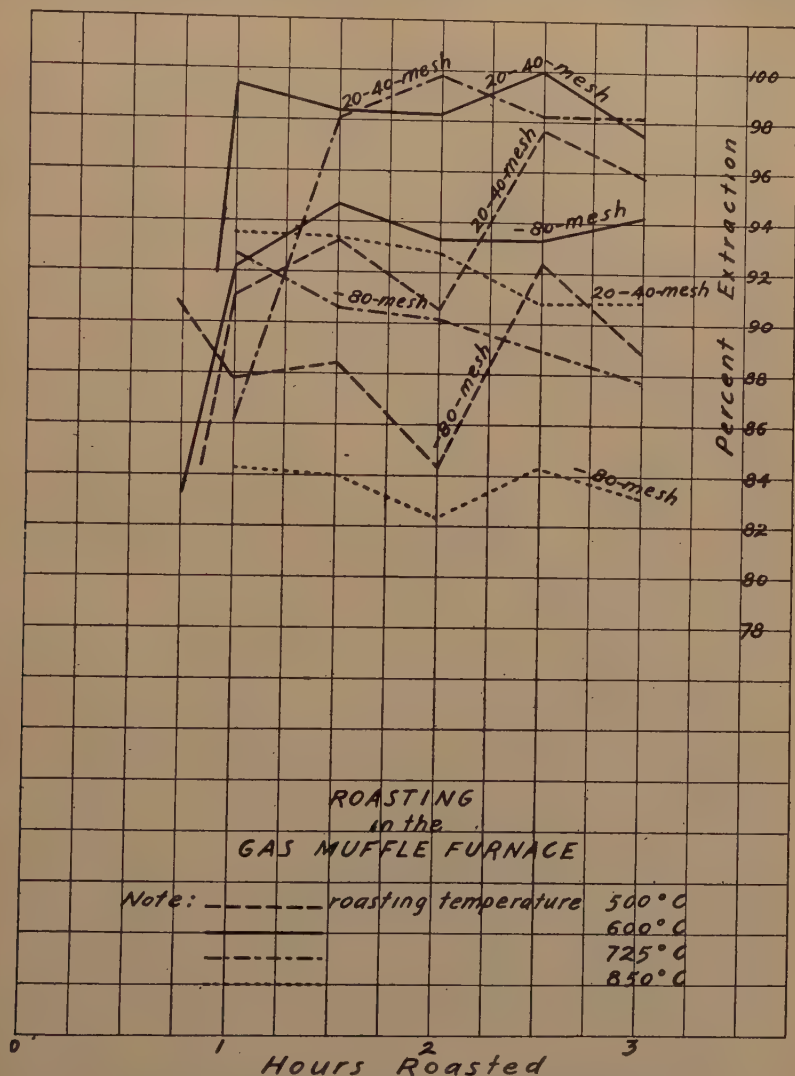


FIG. 5.—ROASTING IN THE GAS MUFFLE FURNACE.

acid diluted to 1,000 cc.). The soluble copper thus obtained is reported as per cent. copper soluble in dilute hydrochloric acid. The sum of these two is reported as the per cent. of total soluble copper. The difference between this and the total per cent. of copper found in the sample

is reported as per cent. insoluble copper. By dividing per cent. of total soluble copper by total per cent. of copper in sample, per cent. extraction was determined.

On referring to Fig. 5, Roasting in the Gas Muffle Furnace, where material through 20 on 40 mesh and material through 80 mesh were



FIG. 6-A.—ROASTING IN THE LARGE GAS FURNACE.

roasted side by side in separate roasting dishes, it will be seen, by leaching with dilute acid, that the material through 20 on 40 mesh always gave a higher extraction than the material through 80 mesh. As regards temperature, the roast conducted at 850° C. gave the poorest extraction, the temperature being too high. (The series of roasts indicates that at temperatures above 725° C. the resulting copper oxide forms insoluble

compounds.) The roast conducted at 500° C. gave better extraction, but, as shown by subsequent roasts, the temperature was too low to secure the best extraction on leaching. Roasts conducted at 600° C. and 725° C. are equally good on material through 20 on 40 mesh; but on material through 80 mesh, the 600° temperature gave material which yielded the

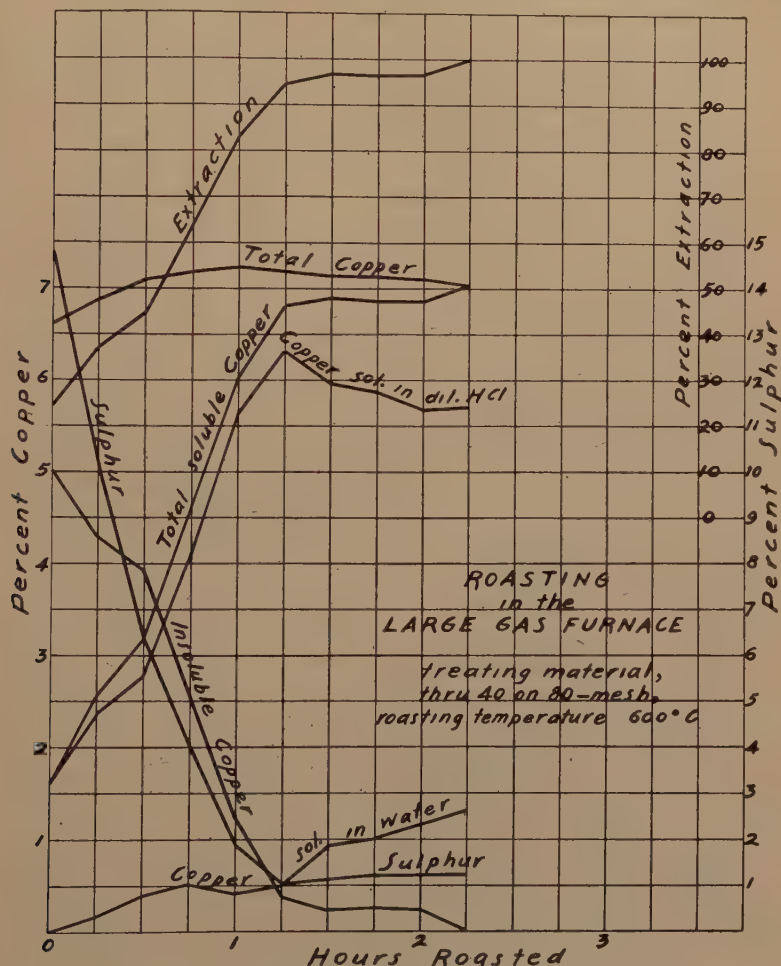


FIG. 6-B.—ROASTING IN THE LARGE GAS FURNACE.

best extraction. Consequently the temperature of 600° C. was selected as the best for subsequent roasting in the large gas roasting furnace.

There was no muffle in the large gas roasting furnace and its operation was similar to that of a gas-fired reverberatory furnace. The hearth, which was removable, consisted of a sheet-iron pan, lined with firebrick (22½ by 9 in.—inside measurement). Two series of roasts were made;

viz., A-Series and B-Series. Eight pounds of material were roasted per charge. The temperature of 600°C . was held constant. The materials roasted were: through 20 on 40 mesh; through 40 on 80 mesh; through 80 mesh; whole through 20 mesh. In all about 100 lb. of ore were roasted. Samples were taken during roasting at $\frac{1}{4}$ -hr. intervals. Of the several

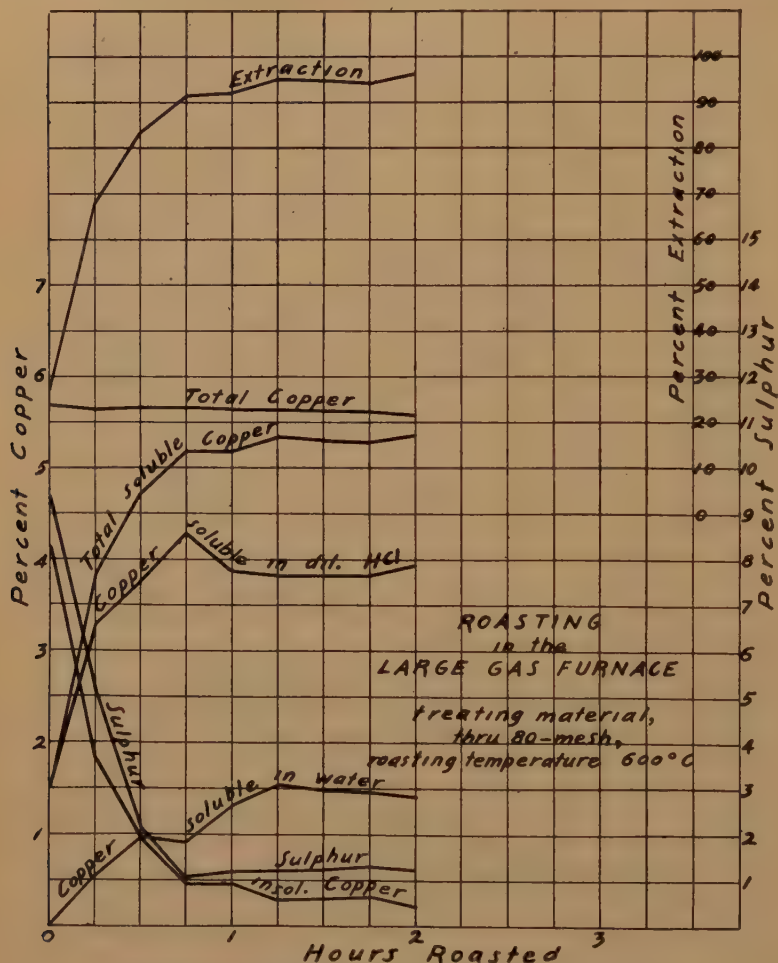


FIG. 6-C.—ROASTING IN THE LARGE GAS FURNACE.

roasts made of one size material, in the A-Series and in the B-Series as through 20 on 40 mesh, average samples were made. These samples were ground through 100 mesh, and tested for soluble copper, in the same manner as the samples resulting from the roasting in the gas muffle furnace.

Referring to Figs. 6-A, 6-B, 6-C, 6-D, Roasting in the Large Gas

Furnace, it will be seen that an extraction of 96 per cent. was secured in all roasts, except on the through 20 on 40 mesh material, which was but 93 per cent. The roasts were all carried on for a longer time than was necessary. An inspection of the curves will enable one to determine the most desirable time of drawing the roasts.

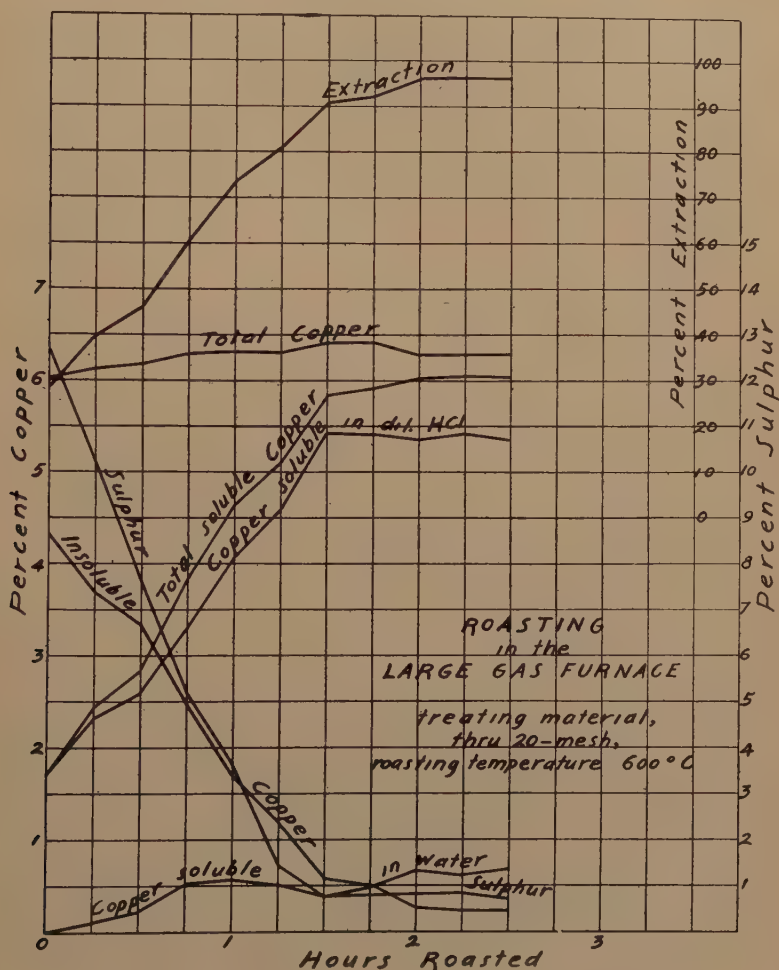


FIG. 6-D.—ROASTING IN THE LARGE GAS FURNACE.

Leaching with Dilute Sulphuric Acid

In order to forecast, as well as could be done in a small way, what might be expected in practice, regarding extraction and acid consumption, the following leaching experiments were made: 20 g. of roasted ore, to-

gether with 200 cc. of 10 per cent. H_2SO_4 ,² were introduced into glass-stoppered bottles, which were continually shaken. Two tests were made, one at a temperature of 21° C., the other at a temperature of 100° C. The material here leached was in the same condition, as regards degree

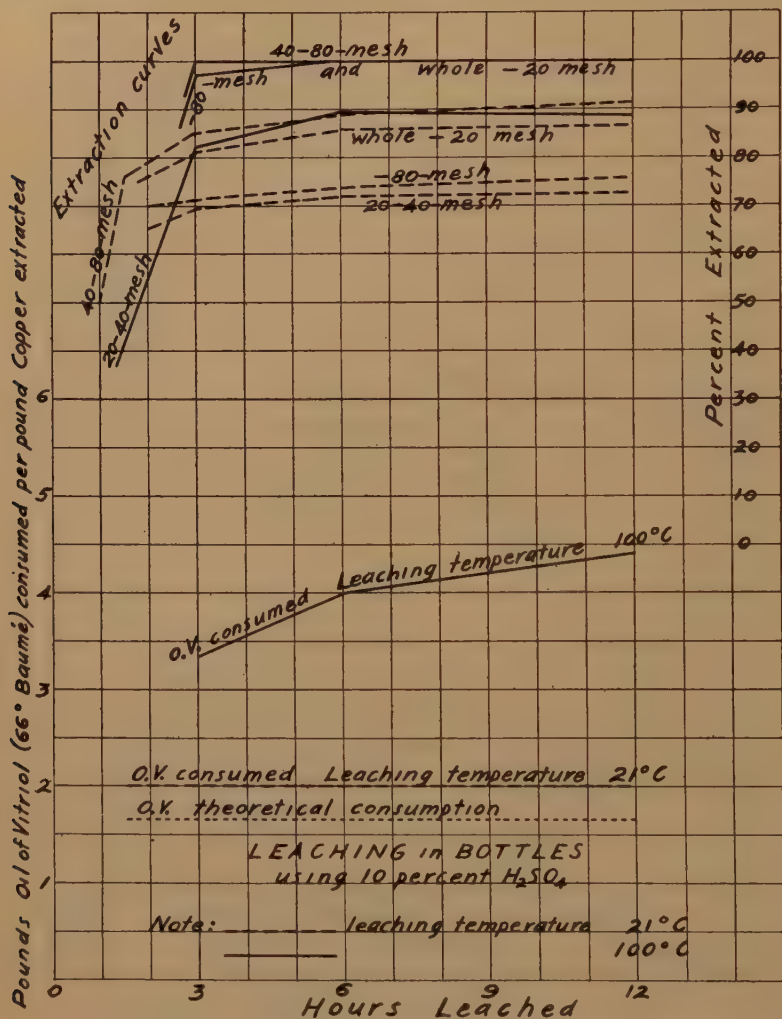


FIG. 7.—LEACHING IN BOTTLES.

of comminution, as when it left the sizing sieve, except as the roasting may have affected it. It is to be noted that it was not so with the samples

² In this paper per cent. of a constituent of a solution signifies grams of the constituent per 100 cc. of the solution: "per cent. H_2SO_4 " signifies grams of free H_2SO_4 per 100 cc.; "per cent. copper," grams of copper per 100 cc.

of roasted materials earlier tested for solubility of the copper contents, and which were all ground through 100 mesh before boiling in the casserole with water and dilute hydrochloric acid.

In the test made at 21° C., the extraction ranged from 72 to 90 per cent. The extraction was not improved by continuing the operation longer than 6 hr. The acid consumption was moderate, being 2.0 lb. of oil of vitriol (66° Bé.) per pound of copper extracted, as compared with 1.65 lb. theoretically required to dissolve 1 lb. of copper existing as oxide. The test which was made at 100° C. gave an extraction of 90 per cent. with the through 20 on 40 mesh material, while with all other materials the extraction was 100 per cent. The acid consumption was somewhat higher, as Fig. 7 shows. The time required for efficient leaching was between 3 and 6 hr. Nothing was gained by increasing the time of leaching beyond 6 hr.

The Electrolytic Precipitation

The Electrolytic Plant.—The electrolytic plant, designed for experimenting in the deposition of copper from solution with insoluble anodes, comprised a motor-generator set, circulating pumps, electrolytic cells, and a testing table. (See Figs. 8, 9, and 10.)

The motor-generator set used was a Robbins & Meyers $\frac{1}{4}$ -hp., 110-volt, 60-cycle, 1,750 r. p. m., single-phase motor, belted to their 0.125-kw., 10-volt, D. C., 1,750 r. p. m., compound-wound generator.

A small acid-proof pump could not be found in the market, so a single-acting acid-proof pump was designed. Four pumps were made. A 2-oz. syringe, No. 135, made by the American Hard Rubber Co., was used as a pump barrel. A block of dry maple was suitably bored. There were two ball valves; the seats were rings of $\frac{1}{4}$ -in. hard rubber; the balls, glass agates. The balls were ground into the seats with emery. The seats entered the cavities bored for them snugly. P. & B. acid-proof paint secured the seats in place so that there was no leakage. The syringe, the suction pipe, and the discharge pipe, entered holes in the maple block prepared for them. Water-tight connections were made by suitable soft-rubber packing rings. The Robbins & Meyers, $\frac{1}{8}$ -hp., 110-volt, 60-cycle, 1,750 r. p. m., single-phase motor, through suitable pulleys and gears operated the four pumps, which made 35 strokes per minute. (See Fig. 10.)

The electrolytic cells used in the experiments of this paper were four cells with one cathode each, two cells with two cathodes each, and two cells with four cathodes each. Each cell had one more anode than cathodes. (See Fig. 9.) Figs. 11 and 12 show one cell with four cathodes, assembled and taken apart. The electrodes were spaced $\frac{3}{4}$ in. from center to center. Anode No. 6, seen in Fig. 13 (lower left-hand corner), was

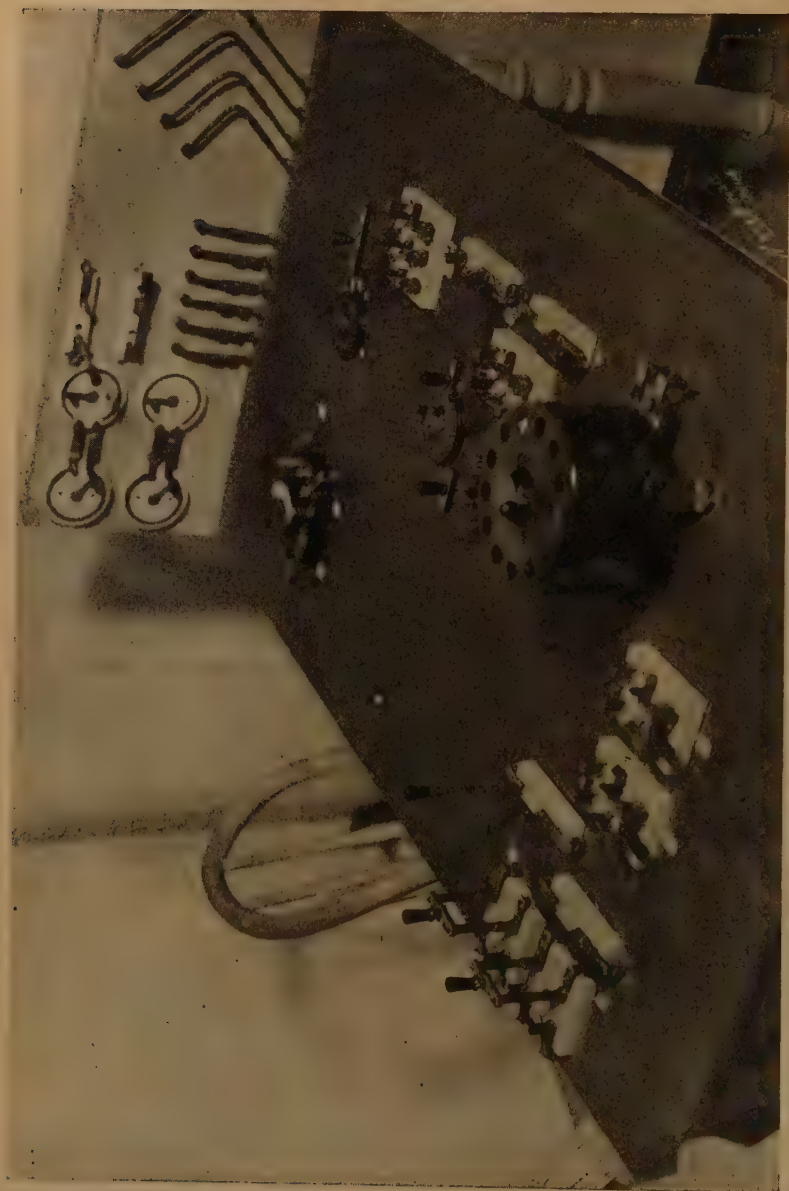


FIG. 8.—TESTING TABLE.

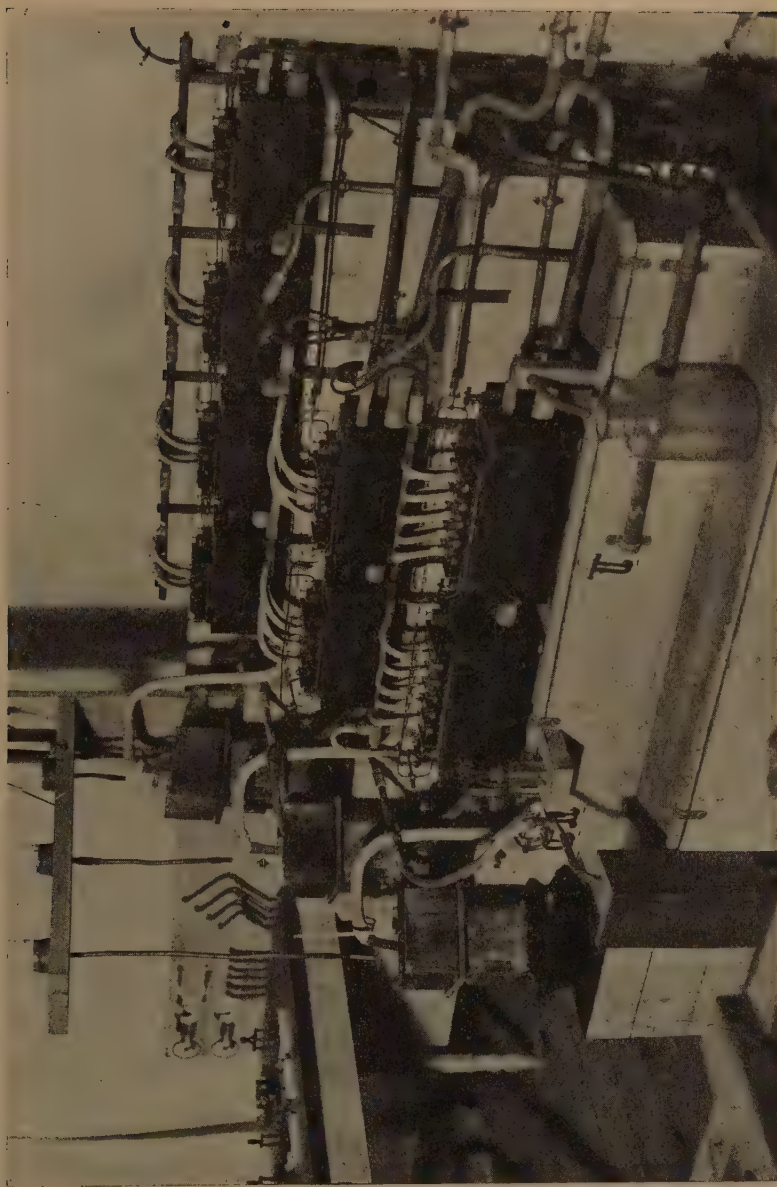


FIG. 9.—ELECTROLYTIC CELLS.

used in Tests Nos. 1 and 2. It was made from a sheet of lead (8 lb. to the square foot), $2\frac{1}{2}$ in. wide and of the proper length. The upper edge was wrapped around a piece of No. 6 bare copper wire by which it was suspended in the cell. Cathodes may likewise be seen in Fig. 13 (middle row). The cathode was made of a sheet of thin copper attached to a copper clamp by which it was suspended in the cell. There were four insulating blocks of hard rubber, 1 by 1 by $\frac{1}{2}$ in., beneath the cell. There were four pieces of hard rubber, 1 by 1 by $\frac{5}{16}$ in., with a $\frac{3}{16}$ -in. upward projection for insulating and holding the bus-bars in place. In Fig. 12, just below the cell, the bus-bars may be seen. The bus-bar was comprised of a 1 by 1 in. brass angle and a 1 by $\frac{1}{4}$ in. hard-rubber strip. The rubber strip was bolted to the vertical leg of the brass angle. There were two pieces of hard rubber spacing pieces used for holding the bus-bars rigidly in place. The method of connecting the electrodes with the bus-bars was similar to practice. The brass and the hard rubber of the bus-bars were so cut that when the horizontal arms of the electrodes rested on them, on one side the arms of the anodes rested on the brass plus conductor while the arms of the cathodes were insulated by hard rubber, and on the other side the arms of the anodes were insulated by hard rubber while the arms of the cathodes rested on the brass minus conductor. The binding posts with No. 6 bare copper wire served for connecting between cells. (See Figs. 9, 11, and 12.)

Quite elaborate piping systems about the cells may be seen in Fig. 9. The elevated temperature tank and the cells of a step were connected by 1-in. lead pipe, the connections to and from the step circulating pump were made by $\frac{1}{2}$ -in. lead pipe. For the progressive circulation of the electrolyte through the plant, $\frac{1}{2}$ -in. lead pipe was used. The sulphur dioxide gas was conveyed to the cells by its system of lead piping. The gas was supplied from a cylinder of liquid gas. The gas cylinder was connected by small rubber tubing to the end of a 2-ft. length of 1-in. lead pipe. From the rear side of this pipe, four $\frac{1}{2}$ -in. lead pipes, leading to different parts of the plant, passed beneath the cells. (See Fig. 9, in foreground.) There was placed horizontally above the cells of each step a $\frac{1}{2}$ -in. lead pipe. This pipe was fitted with $\frac{1}{4}$ -in. tee connections which were connected to the hollow anodes, used in Tests Nos. 3 and 4, by rubber tubing. Bottles were introduced between the ends of the four distributing pipes and these horizontal pipes, just mentioned, in order to indicate the amount of gas flowing. The four indicator bottles for judging the rate of flow of the gas contained a little water through which the gas bubbled. A screw clamp on the rubber tubing connection, between each distributing pipe and its indicator bottle, adjusted the distribution of gas.

A poplar kitchen table, with top 30 in. by 48 in., was selected for the testing table. A Weston miniature, precision, direct-current volt-am-

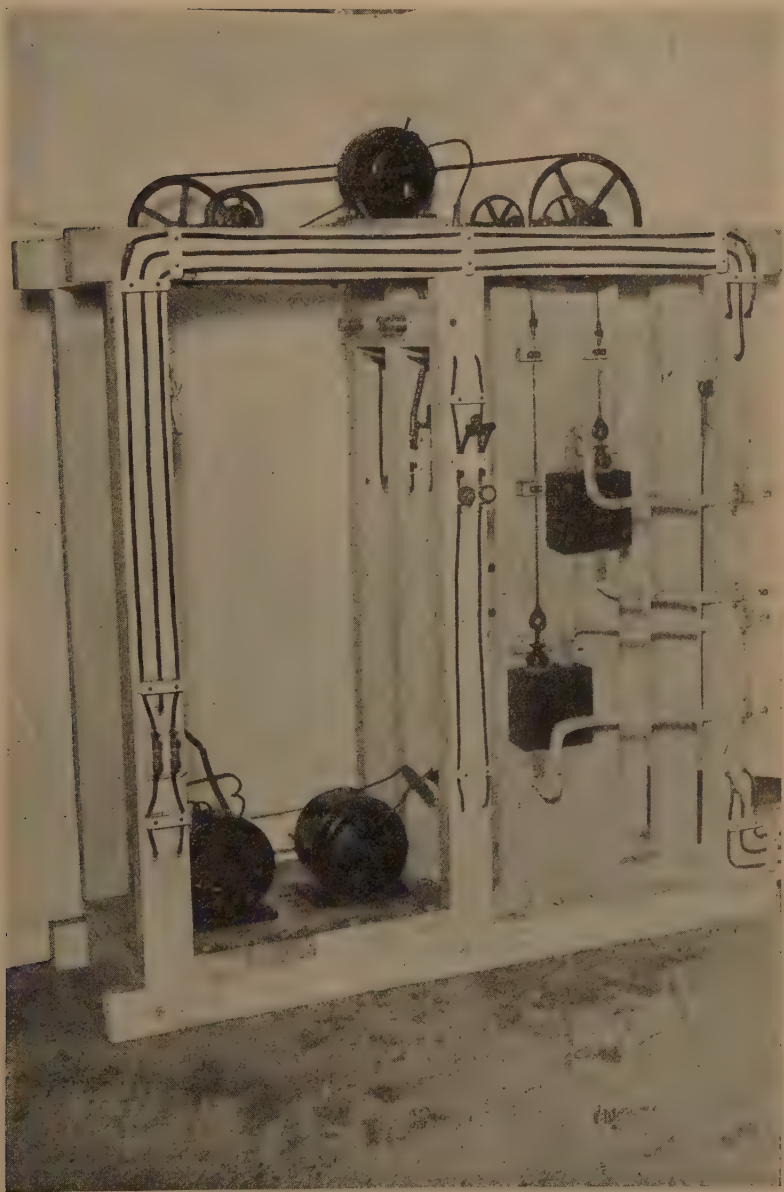


FIG. 10.—FRAME.

meter was used (Model 280 triple-range portable volt-ammeter, Weston Electrical Instrument Co., Newark, N. J., Bulletin No. 8, 1912). The scales were: 150, 15, 3 volts; 30, 15, 3 amperes. The instrument was permanently placed and connected on the table top at the operator's left hand. The table was wired and connected with the rest of the plant, so that all the readings desired might be taken on the one instrument when the switches were properly manipulated. The following readings could be taken:

1. Total amperes, load on generator.
2. Amperes, No. 1 load circuit. (Plant could be operated as one circuit or as two parallel circuits.)



FIG. 11.—CELL ASSEMBLED.

3. Amperes, No. 2 load circuit. (Plant could be operated as one circuit or as two parallel circuits.)
4. Generator volts.
5. Individual volts across cells.

There were fourteen voltage wires, each terminating at one end (the cell end) in a tee connector. In the foreground of Fig. 9 may be seen the cell end of one voltage wire (No. 12). In connecting up the cells (arrangements might be varied in different tests), there was always to be found close to the cell the cell end of a voltage wire. In making series connections between consecutive cells, No. 6 copper connecting wires from the respective binding posts of the bus-bars of the two cells entered a tee connector, one leg of which was permanently attached to the end (cell

end) of a voltage wire. All the fourteen voltage wires passed to the testing table where they connected to two 13-point circular switches. The switch to the operator's left had connected to it wires Nos. 1 to 13, while the switch to his right had connected to it wires Nos. 2 to 14. On properly setting the two 13-point switches and throwing the three-pole double-throw switch adjoining, the individual volts across cells, or the

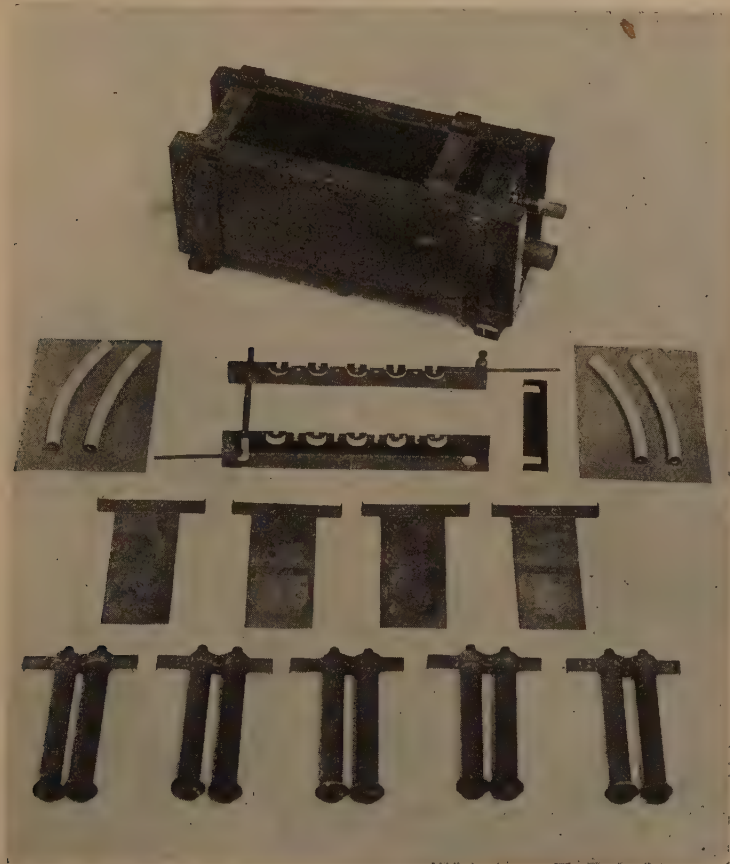


FIG. 12.—CELL TAKEN APART.

sum voltage of any number of cells, could be recorded by the instrument on suitable scale. Not all the cell ends of the voltage wires were connected in any one test. When the plant was wired up for some one test, the voltage wires in use were noted and the corresponding switch points on the table top were tagged for the operator's use. All switch points connected with unused voltage wires were dead.

The generator field rheostat is seen in the foreground of Fig. 8. To the operator's left, on the vertical panel, were four battery rheostats, con-

nected in parallel, serving as a variable load in series with the cells in load circuit No. 2. These battery rheostats could be short circuited, when the generator field rheostat would give the desired voltage regulation. When a lower voltage was desired than the field rheostat could give, then the variable resistance in series was used.



Fig. 13.—ELECTRODES.

Test No. 1.—Arrangement A, in "Example of Step Arrangement of Plant," was employed, operating No. 1 Step and No. 2 Step and omitting the Finishing Step. No. 1 Step had four cells, connected in series, each with one cathode. No. 2 Step had two cells, connected in parallel, each with two cathodes: this combination will be spoken of as one cell of four

TABLE I. TESTS Nos. 1, 2, 4.

Test No.	Copper Solution Feed		No. 1 Step				No. 2 Step				Finishing Step.				Duration		No. 1 Step										No. 2 Step																			
	c.c. per 12 hours	Cu-percent	Composition of electrolyte.		Step circulation - c.c. per minute	Composition of electrolyte.		Step circulation - c.c. per minute	Volume of solution supplied to step at beginning - c.c.	Volume of water added to step during test to compensate for evaporation - c.c.	Cu-percent	H ₂ O (free) percent	Cu-percent	H ₂ O (free) percent	Cu-percent	H ₂ O (free) percent	Cu-percent	H ₂ O (free) percent	Amperes	Cathode surface per cell - sq. ft.	Current density - amperes per sq. ft.	Average volts per cell	Number of cells in series	Theoretical Cu deposited - grams	Actual Cu deposited - grams	Total Kw. hours	Current efficiency - percent	Cu deposited - lb. per kw. hour																		
			at beginning	at end		at beginning	at end																						at beginning	at end	at beginning	at end	at beginning	at end	at beginning	at end	at beginning	at end	at beginning	at end						
1	1072	6.0	0.0	4960	1700	1400	1500	6.93	1.52	6.90	3960	935	1400	0.235	0.866	0.993	8.61	5210	1170	1400	0.743	15.40	0.049	15.80	12	12	1648	0.024	20	1.80	4	93.6	0.639	0.143	90.80	1.32	0.326	5	1.70	1	23.4	20.95	0.034	93.70	1.44	1.44
2	3042	3.7	0.0	4360			1400	1.50	6.24	1.49	3950	1400	0.243	15.40	0.740	15.10	5210	1170	1400	0.743	15.40	0.049	15.80	12	12	1648	0.024	20	1.80	4	93.6	0.639	0.143	92.00	1.34	0.326	5	1.70	1	23.4	20.95	0.034	93.30	1.43	1.43	
4	3042	3.7	0.0	4360			1400	1.50	6.24	1.57	3950	1400	0.243	15.40	0.742	15.70	5210	1170	1400	0.743	15.40	0.049	15.80	12	12	1648	0.024	20	1.30	4	93.6	0.639	0.103	91.80	1.84	0.326	5	1.00	1	23.4	22.64	0.008	95.80	2.50	2.50	

Note : For electrical record of Finishing Step , Tests No.2 and 4 see Fig.15

cathodes. Note that the plant was operated with one-third as many cells as are indicated in the diagram, Fig. 16 (Example of Step Arrangement of Plant—A). Evaporation was compensated for by adding sufficient water to the temperature tank of No. 1 Step and to the temperature tank of No. 2 Step. The relative amounts of water for each step were proportional to the step surface exposed to evaporation. Consequently the overflow to the sump of No. 2 Step was equal in volume to that of the copper solution fed to the temperature tank of No. 1 Step. (See Table I.)

Current density of 20 amperes per square foot was selected for No. 1 Step. The area (sum of the two sides) of the cathode is 0.0824 sq. ft., therefore a current of 1.648 was supplied. The volume of copper feed solution required for 12 hr., in order to supply the cells of No. 1 Step with an amount of copper equal to that deposited in that step by the current selected plus the copper carried on by the progressive circulation so that the electrolyte would remain constant in composition, was determined as follows:

$$\frac{1.648 \times 4 \times 12 \times 1.1855 \times 0.90}{0.75 \times 0.06} = 1,872 \text{ c. c. copper feed solution required for 12 hr.}$$

Where,

Current amperes.....	1.648
Number of cells in series, No. 1 Step.....	4
Duration, hours.....	12
Theoretical copper deposited by 1 ampere-hour, grams.....	1.1855
Current efficiency, per cent., assumed.....	90
Copper deposited in No. 1 Step, expressed as per cent. of that in feed.....	75
Copper in feed solution, per cent.....	6

1,872 c. c. of No. 1 Step electrolyte, containing 1.5 per cent. copper, overflowed from the last cell of No. 1 Step to the temperature tank of No. 2 Step. This supplied No. 2 Step with an amount of copper equal to that deposited by the current in that step, plus the copper carried on to the sump by the progressive circulation, as the following calculation shows:

$$\frac{1.648 \times 1 \times 12 \times 1.1855 \times 0.90}{0.75 \times 0.015} = 1,872 \text{ c. c., overflow of No. 1 Step electrolyte, required for No. 2 Step.}$$

Where,

Number of cells in series, No. 2 Step.....	1
Copper in overflow of No. 1 Step electrolyte, per cent.....	1.5

The strength of the electrolyte in H_2SO_4 , which is developed by the acid liberated by the copper on deposition, was determined as follows:

No. 1 Step.

$$0.06 \times 0.75 \times 1.54 = 0.069 \text{ (or 6.93 per cent.)}$$

Where,

Copper in feed solution, per cent.....	6
Copper deposited in No. 1 Step, expressed as per cent. of that in feed.....	75
H ₂ SO ₄ freed per unit of copper deposited.....	1.54

No. 2 Step.

$$0.06 \times 0.9375 \times 1.54 = 0.0866 \text{ (or 8.66 per cent.)}$$

Where,

Copper deposited in Nos. 1 and 2 Steps, expressed as per cent. of that in feed.....	93.75
--	-------

In starting up the plant, each step was supplied with the necessary volume of electrolyte to put it in operation. The composition of this initial charge of electrolyte in copper and acid was that which the above calculations indicate and which may be seen in Table I. The results of this test show that when employing the Step Arrangement of Plant, and when adding water to each step to compensate for evaporation, the initial values of copper and acid contents of the electrolyte remain constant during the electrolysis. The results of this test also show the performance when copper is deposited, using insoluble lead anodes. (See Fig. 9; Tables I and II; and Fig. 16, Diagram Example of Step Arrangement of Plant—A.)

Test No. 2.—Arrangement A, in "Example of Step Arrangement of Plant," was employed, using No. 1 Step, No. 2 Step, and the Finishing Step. The arrangement of the cells in Nos. 1 and 2 Steps was the same as in Test No. 1. The Finishing Step had two cells, connected in parallel, each with four cathodes; this combination will be spoken of as one cell of eight cathodes. The operation of the Finishing Step of this test will be discussed in Test No. 4.

In order to simplify the operation, a weaker copper feed solution was made for No. 1 Step by adding 1,170 c. c. water to 1,872 c. c. of 6.0 per cent. copper solution, producing 3,042 c. c. of feed solution containing 3.7 per cent. copper, which was used for the 12-hr. run. This supplied No. 1 Step with the same amount of copper as in the previous test and with the water necessary to compensate for evaporation. There should have overflowed from Step No. 1, during the 12-hr. run, 1,872 c. c. of solution containing 1.5 per cent. of copper and 6.93 per cent. of H₂SO₄. When, as in Test No. 1, evaporation in No. 2 Step is compensated for by the addition of an equivalent amount of water, this becomes the step electrolyte with 0.375 per cent. of copper and 8.66 per cent. of H₂SO₄. But in this test no water was added to No. 2 Step, so assuming evaporation to be the

same in amount as during the preceding test, namely 925 cc., there should have overflowed to the sump $1,872 - 925$, or 947 cc. solution of the following composition:

$$0.00375 \times \frac{1872}{947} = 0.00743 \text{ (or 0.743 per cent.) copper.}$$

$$0.0866 \times \frac{1872}{947} = 0.171 \text{ (or 17.1 per cent.) } \text{H}_2\text{SO}_4.$$

Note that in Test No. 2, as well as in Test No. 4, the initial charge of electrolyte to all steps deviated slightly in H_2SO_4 composition from the value which was calculated that it should have been. There was charged to No. 1 Step, a solution containing 6.24 per cent. H_2SO_4 , instead of 6.93 per cent.; to No. 2 Step and to Finishing Step, 15.4 per cent. H_2SO_4 , instead of 17.1 per cent.

The results of this test show that when employing the Step Arrangement of Plant as outlined above, and which differs in some details from the method used in Test No. 1, and when supplying initial charges of electrolyte of the compositions stated, these initial values remain constant in both copper and acid. The results of this test also show the performance when copper is deposited, using insoluble lead anodes. (See Figs. 9, 15, and 16; Tables I and II.)

Test No. 3.—But one electrolytic cell was used in this test. The intention was to introduce sulphur dioxide gas into the electrolyte during electrolysis in all subsequent work, so this test was run as a preliminary test to determine the best method of introducing the gas as well as the most suitable anode, in order to secure the best depolarizing effect.

Kinds of Anodes.—See Fig. 13: Anodes Nos. 1, 2, 3, 4 and 5, top row, counting from left to right; Anodes No. 6, lower left corner; No. 7, lower right corner.

Anode No. 1 was the lead Anode No. 6, used in Tests Nos. 1 and 2, so modified that SO_2 could be introduced into the electrolyte. Sufficient width and length was cut from one side and the bottom of Anode No. 6, as formerly used, so as to permit of attaching a $\frac{1}{4}$ -in. lead pipe, without increasing its size. This lead pipe was closed at the immersed end, and had its horizontal leg perforated on the upper side with three No. 50 drill holes (about $\frac{1}{16}$ -in. diam.), equally spaced. It was thought that the placing of the perforations thus would give an even distribution of the gas over the surface of the anode. (See Fig. 13, Anode No. 1.)

Anode No. 3 was made of lead pipe to imitate in form the carbon anode (No. 7). The main part is a piece of a $\frac{1}{2}$ -in. lead pipe. The bottom end was closed with a disk of lead in which was drilled a $\frac{1}{8}$ -in. hole (the same size as the central channel in the carbon anode). A short piece of $\frac{1}{4}$ -in. lead pipe was forced into and burned to the upper end of the $\frac{1}{2}$ -in. lead pipe. To this $\frac{1}{4}$ -in. lead pipe rubber tubing was attached

for supplying the sulphur dioxide gas. A soft rubber washer was placed on the pipe, near its lower extremity, to act as an insulator for preventing the occurrence of short circuits in the cell. Such a rubber washer was similarly placed on all other round anodes. This soft rubber washer had the same office as the hard rubber pegs of the flat lead anode (No. 6). The complete anode comprises a pair of these lead pipes which are rigidly held in a special copper clamp. (See Fig. 13, Complete Anode No. 2.)

In Anode No. 3, a $\frac{1}{4}$ -in. lead pipe extends through a short section of a $\frac{1}{2}$ -in. lead pipe, which enables it to be held firmly in the standard copper clamp. A piece of charcoal free from flaws was cut exactly $\frac{3}{4}$ in. square by $2\frac{1}{2}$ in. long, and bored so as to fit snugly over the $\frac{1}{4}$ -in. lead pipe. The sulphur dioxide gas enters the electrolyte from the bottom of the $\frac{1}{4}$ -in. lead pipe. (See Fig. 13, Anode No. 3.)

Anode No. 4 is of carbon. The carbon tube from which the anode was made is $\frac{3}{4}$ in. outside diameter, $\frac{1}{8}$ in. inside diameter and 12 in. long. This is an electric-arc carbon, incomplete in manufacture, and made by the National Carbon Co. Were it completed, the center channel would be filled with another grade of material, and then it would be a standard carbon for use in an electric arc for industrial purpose. This anode had a number of small holes made with a No. 50 drill, entering radially to the center channel. (See Fig. 13, Anode No. 4.)

Anode No. 5 differs from Anode No. 4 in that saw cuts were made instead of drill holes. These saw cuts extend half way through the carbon tube intersecting the center channel. (See Fig. 13, Anode No. 5.)

Anode No. 7 differed only from Nos. 4 and 5 in that it had no lateral perforations, the gas entering the electrolyte from the bottom extremity of the center channel. (See Fig. 13, Anode No. 7.)

The Depolarizing Effect of Sulphur Dioxide Gas.—On the assumption that all chemical energy is transformed completely into electrical energy, it was found by calculation that when no depolarizer is used, in a copper sulphate electrolyte, the theoretical counter E. M. F. of polarization is 1.22 volts, and when sulphur dioxide is used as depolarizer, the theoretical counter E. M. F. of polarization is -0.15 volt.³ Thus, theoretically, sulphur dioxide gas should reduce the polarization by 1.37 volts, changing the polarization of the cell from 1.22 volts opposing the generator to 0.15 volt, pulling with the generator. Consequently in this experiment a reduction of polarization of 1.37 volts has been taken as the standard of perfection. In the calculation of the counter E. M. F. of polarization, the correction term of absolute temperature times the temperature coefficient of the electrical energy was neglected, because its value is unknown. The results obtained were somewhat influenced by the omission of this correction term.

Polarization volts were determined as follows: During the test, when

³ *Transactions of the American Electrochemical Society*, vol. xxv, p. 230 (1914).

running at the current density desired, Reading 1 was taken of amperes and volts. By suitably manipulating the generator field rheostat, together with the series resistance in the circuit, the current was reduced appreciably. New current value, together with corresponding volts, was quickly recorded, giving Reading 2. The current was quickly brought back to its running value when Reading 3 was again taken of amperes and volts. Reading 3 should correspond to Reading 1. If it did not, the system was allowed to assume again its normal constant conditions and the same operation was repeated. All the readings were taken in a few seconds.

Example illustrating determination of polarization:

Reading 1, at C. D. 8 amperes per sq. ft.,

$$I_1 = 2.637 \text{ amperes; } V_1 = 0.710 \text{ volt.}$$

Reading 2,

$$I_2 = 1.000 \text{ amperes; } V_2 = 0.533 \text{ volt.}$$

$$I_1 - I_2 = 1.637 \text{ amperes; } V_1 - V_2 = 0.177 \text{ volt.}$$

Then

$$V_1 = E + I_1 R$$

$$V_2 = E + I_2 R$$

$$V_1 - V_2 = (I_1 - I_2) R$$

$$R = \frac{V_1 - V_2}{I_1 - I_2} = \frac{0.177}{1.637} = 0.108 \text{ ohm.}$$

$$E = V_1 - I_1 R = 0.710 - (2.637 \times 0.108) = 0.425 \text{ volt.}$$

R equals ohmic resistance, made up of resistance in the solid conductor, contact resistance, and resistance in the electrolyte. This is constant.

E equals polarization voltage, constant at any one current density. It is assumed that during the time of taking Readings 1, 2, 3, E does not vary. This is proved when Reading 3 checks Reading 1. Results were not recorded unless this was the case.

On trying out the Anodes Nos. 1, 2, 3, 4, 5, 7, which were supplied with sulphur dioxide gas, it was found that with Anodes Nos. 1, 2, 3 there was no reduction in polarization at all. The reason Anode No. 3 was tried was because, from a few random tests with the carbon anode, it had been learned that with this anode there would be beneficial reduction in polarization by the introduction of sulphur dioxide gas. It was thought that charcoal, being quite porous, might promote anode efficiency on account of its known property of occluding gases. A charcoal rod, when used as anode, was found to have too high resistance to permit current to pass, therefore the combined lead-charcoal anode (No. 3) was made. This anode did permit current to pass, but, contrary to expectation, there was no beneficial depolarization.

The carbon anodes Nos. 4 and 5, the former with radial holes and the latter with saw cuts, were tried with their submerged ends of central channel open. Then they were tried with their submerged ends plugged, forcing all the gas entering the electrolyte out through the radial holes and

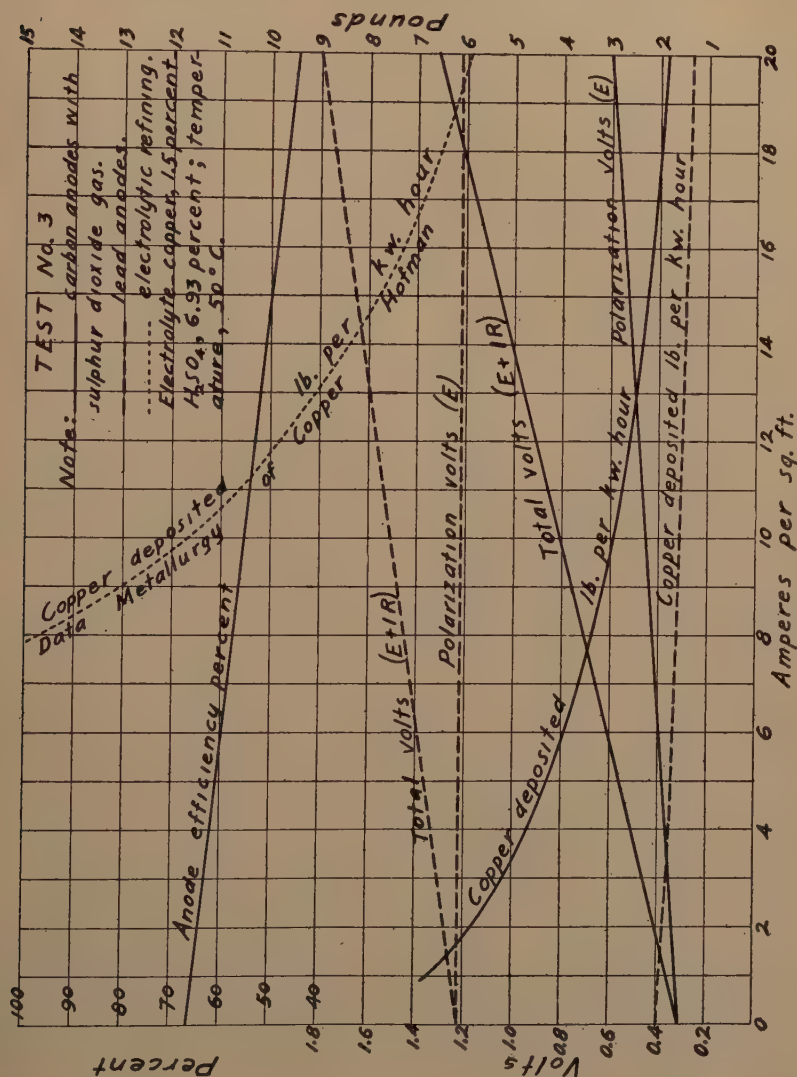


FIG. 14.—TEST No. 3.

saw cuts. A variety of arrangements of holes and saw cuts were tried. Finally Anode No. 7, carbon anode with central channel extending through (not plugged) and with no other opening (neither radial drill holes nor saw cuts), was tried. The gas was delivered into the electrolyte entirely

from the lower extremity of the anode. The depolarization efficiencies of Anodes Nos. 4, 5, 7 were equally good compared one with the other. Therefore Anode No. 7, being the simplest in form as well as one of the most efficient, was selected to be used in Test No. 3 and in Test No. 4. Readings were then taken from which Fig. 14 has been drawn. It is to be noted that during this test, sulphur dioxide gas was admitted through every anode, at such a rate that there was a gentle bubbling of the gas in the electrolyte.

Depolarization was not always equally good with the same carbon anode. It was generally only with a new carbon anode, when on closing the circuit permitting current to pass at current density of 20 amperes per square foot, that a total voltage reading as low as 0.60 volt might be recorded. This extremely low voltage always rose to some higher value which remained constant during the test. A subsequent test with the same anode generally gave the first voltage reading higher, likewise, the reading of the constant value higher. When depositing copper with a current density of 20 amperes per square foot, at one stage of the experimenting it appeared that the total voltage could be held at 0.9 volt, at a later stage, 1.2 volts, while Test No. 4 recorded 1.3 total volts. Apparently a new anode, when first put into use, is more efficient in reducing polarization than subsequently. This may partially account for variations in tests on the same anode. From time to time there was considerable variation in the ohmic resistance of cell (due to contacts), which may account for some of this variation in total volts.

Finally, after completing all the regular readings for the curve sheet, a little further experimenting was done. Ten anodes were used in a test. Gas was admitted by the ten anodes, by four anodes only, and by six anodes only. The reduction in polarization was as good when admitting gas by four anodes or by six anodes as when admitting gas by all ten anodes. The only difference noticed was that the gas was required to flow through the anodes admitting gas somewhat faster than before—probably the volume of gas entering was no more.

In Test No. 3, when using different kinds of anodes, beneficial reduction of polarization was only secured when carbon anodes were used, indicating that the nature of the anode is important. When using the carbon anode, equally good results were secured with all forms of the carbon tube (National Carbon Co. tube). It mattered not whether they were perforated by drill holes or saw cuts, for admitting and distributing the gas to the electrolyte, or whether it entered by every anode. All that was necessary was that the anode should be a carbon tube, with sufficient sulphur dioxide gas admitted to the electrolyte in some way. It appears that the quantity of sulphur dioxide gas required is that necessary to keep the electrolyte saturated with the gas.

Solubility of Sulphur Dioxide in Water

H ₂ O Degrees C	20	30	40	50	60	70	80	90	100
SO ₂ per cent. dissolved..	8.6	7.4	6.1	4.9	3.7	2.6	1.7	0.9	0.0

(H. O. Hofman, *General Metallurgy*, p. 880.)

Referring to the Tossizza patent, which states: "I have thought to use . . . insoluble anodes kept in contact with sulphurous acid and to utilize the known depolarization properties of the said sulphurous acid;" "These anodes can be made of carbon." From the above quotations, together with the results obtained in this test, I infer that Tossizza experimented only with carbon anodes. (See Figs. 8, 9, 10, and 14.)

Test No. 4.—The operation of this test differed from Test No. 2 only in that carbon tube anodes were used in place of the lead anodes, and sulphur dioxide gas was introduced into the electrolyte. Were depolarization perfect in operation, then for each equivalent of copper deposited, two equivalents of H₂SO₄ would appear instead of one when sulphur dioxide gas is not used.⁴ In this test, the initial charges of electrolyte to the different steps had the degree of acidity which had been determined suitable for Tests Nos. 1 and 2 where SO₂ gas was not employed. Consequently, when using sulphur dioxide gas as depolarizer, the electrolytes became more acid in proportion to the anode efficiency as the electrolysis progressed. The tabulation of Test No. 4 shows a decided increase in acidity. (See Table I.) When the anode efficiency is known and when the initial electrolyte charged to the cells is made correspondingly more acid, then the electrolyte should remain constant in acid contents. Anode efficiency is discussed fully in Test No. 3. It was there estimated by percentage reduction in polarization rather than by increased acidity, as with a fine electrical instrument this method was thought to be more accurate than analytical methods. By the electrical method, results were recorded instantaneously, while the determination of the sulphuric acid by chemical analysis would require days to secure a series of results.

Referring to the tabulation of Test No. 4, where the average results of readings taken half-hourly are recorded, quite a saving in power in Nos. 1 and 2 Steps of Test No. 4 over Test No. 2 will be seen.

Finishing Step.—The curve sheet of the Finishing Step (Fig. 15) represents the performance of this step of the process in Tests Nos. 2 and 4.

The 5,210 cc. of electrolyte, containing 0.743 per cent. copper and 15.4 per cent. H₂SO₄, with current of 1.648 amperes passing, required theoretically, 19.85 hr. to completely deposit the total copper contents. In these tests, when carried to the 19.85-hr. point, the results were:

⁴ *Transactions of the American Electrochemical Society*, vol. xxv, p. 230 (1914).

	Test No. 2	Test No. 4
Copper deposited, per cent. of that supplied to Finishing Step.....	93.5	90.2
Average current efficiency.....	93.5	90.2

While the extraction is high and current efficiency good in both tests, it would not be desirable to carry the electrolysis to this point (19.85-hr.

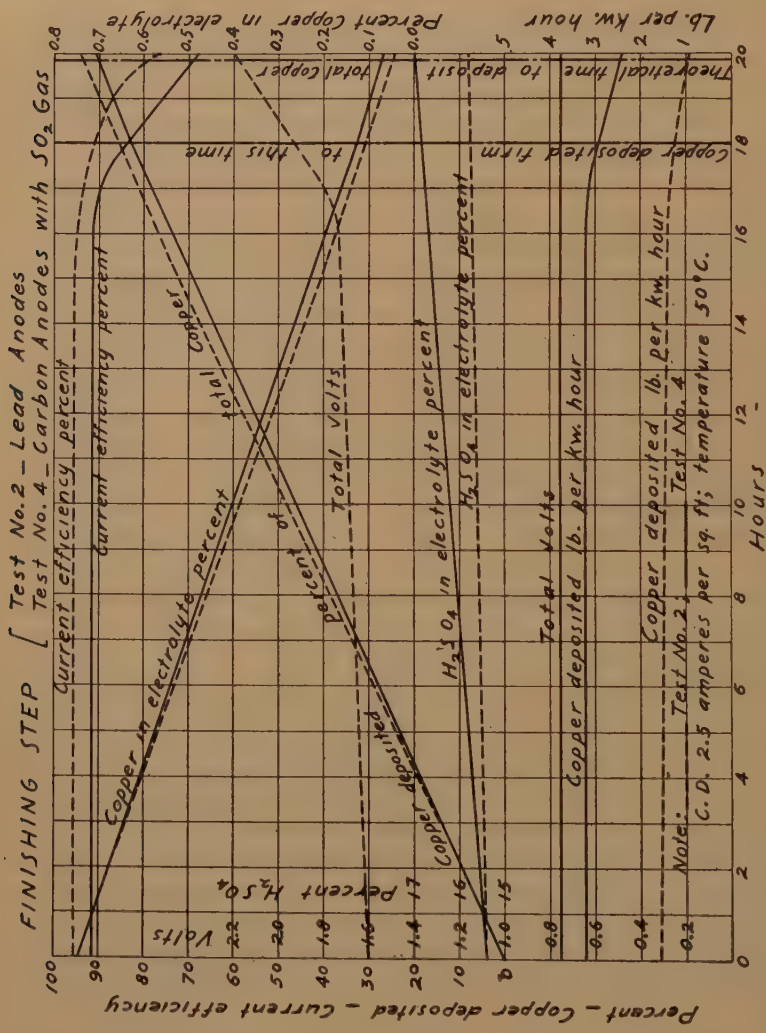


FIG. 15.—FINISHING STEP.

point) in Test No. 2, because the last of the copper comes down spongy and falls to the bottom of the cell. In Test No. 4, the electrolysis may be carried to the 19.85-hr. point and a good cathode produced on account of the presence of SO₂ in the electrolyte. But even in Test No. 4, this point is the limit of feasible electrolysis. At the 20-hr. point, great

blotches of sulphide of copper formed on the cathode. Test No. 2 may be carried to the 18th hour and produce a good, firm deposit of copper. In Test No. 2, when carried to the 18-hr. point, the results were:

Test No. 2

Copper deposited, per cent. of that supplied to Finishing Step.....	85.5
Average current efficiency.....	94.4

Note that there is a decided saving in power by the use of sulphur dioxide gas with the carbon anodes, in the Finishing Step of Test No. 4. In Test No. 4, there is no rise of polarization as the copper contents becomes depleted, while the total voltage becomes 2.2 volts in Test No. 2.

The middle row of Fig. 13 consists of six cathodes. Counting from left to right, the first group of three (Nos. 1, 2, 3) was used in Tests Nos. 1 and 2; the second group of three (Nos. 4, 5, 6) was used in Test No. 4. Immediately below Cathode No. 2, is one of the lead anodes used in Tests Nos. 1 and 2; immediately below Cathode No. 5, is one of the carbon anodes used in Test No. 4. Cathodes Nos. 1 and 4 are from No. 1 Step of their respective tests; Cathodes Nos. 2 and 5 are from No. 2 Step of their respective tests; and Cathodes Nos. 3 and 6 are from the Finishing Step of their respective tests.

The performance in Test No. 4, in pounds of copper deposited per kilowatt-hour—between 1.82 and 3.7 lb.—is not equal to that in Test No. 3. Owing to the construction, electrodes light in weight, and resting on bus-bars, the contact resistance was abnormally high. Special precaution was taken in Test No. 3 largely to eliminate the contact resistance. Even in Test No. 3, the ohmic resistance is to that of practice as 0.108 is to 0.062. Consequently it is believed that even the showing of Test No. 3 can be bettered in practice, and with this plant on repetition.

Test No. 1 showed the performance of lead anodes when used with the step system. Test No. 2 was a modification of Test No. 1, seeking to place the operation on a more practical basis. Test No. 3 compared the performance of different kinds of insoluble anodes with and without sulphur dioxide gas. Test No. 4 showed the performance of carbon tube anodes with sulphur dioxide gas introduced into the electrolyte, when used with the step system. The operation of the Finishing Step with SO_2 gas showed a remarkable saving of power as well as a better deposit of copper over that of the Finishing Step in Test No. 2 with lead anodes and no gas. (See Figs. 9, 13, and 15; Table I.)

Step System of Electrolytic Precipitation.—The copper solution used in the preceding experiments was obtained by dissolving copper sulphate in water. The electrolytic plant was not supplied with copper solution obtained by leaching the ore, because the amount of ore roasted was insufficient to keep the electrolytic plant in operation for the length of time desired. Moreover, it is believed that the only additional informa-

TABLE II.—*Example of Step Arrangement of Plant—Arrangement A*
Copper Contents of the Electrolyte of the Steps held in Geometrical Ratio $\frac{1}{4}$ (except the Finishing Step)

	Feed- Copper Solution	No. 1 Step	No. 2 Step	No. 3 Step	No. 4 Step	Finishing Step
Relative copper contents of electrolyte...	1	$\frac{1}{4}$	$\frac{1}{16}$	$\frac{1}{64}$ —0
Copper contents of electrolyte, per cent....	6.0	1.5	0.375	0.375—0
H ₂ SO ₄ contents of electrolyte, per cent....	0	6.93	8.66	8.22—9.24
Copper remaining in electrolyte, expressed as per cent. of that in feed.....		25	6.25	0
Copper deposited, expressed as per cent. of that in feed.....		75	18.75 (Nos. 1 and 2 Steps)	6.25
Total copper deposited, expressed as per cent. of that in feed.....		75	93.75	(All Steps) 100
Number of cells in series in each step.....		x	$\frac{x}{4}$	x
Relative electrode area per cell.....		1	4	12
Relative C. D. in each step.....		1	$\frac{1}{4}$	$\frac{1}{16}$

N. B.—When using carbon anodes with sulphur dioxide gas, the acid contents will be higher than given in these tables.)

These tables were worked up, assuming that there was no evaporation of the electrolyte during its passage through the plant, or, what amounts to the same thing, that water was added to the different steps equal in amount to the evaporation occurring in the different steps.

tion that could have been obtained by treating copper solution resulting from ore leaching, would have been the effect of accumulation of impurities in the electrolyte. In order to determine the effect of accumulation of impurities in the electrolyte, and to inaugurate the necessary purification methods, such as chemically purifying the electrolyte or wasting a sufficient amount of barren solution, a greater amount of material than was at hand would have been required, as well as a long campaign of operation. Much information is available regarding the maintaining of the purity of solutions in a cyclic process. The analysis of the ore together with the results of the leaching tests, and the acid consumption, enables one to judge the quantity of impurities entering the leaching solution in the treatment of the ore now under discussion.

The electrolysis of the copper sulphate solution was conducted in steps. All the electrolytic cells were connected in series, while the electrodes of the individual cells were connected in parallel. Each step maintained constant the composition of the electrolyte—both in copper and acid.

The cells comprising each step have two circulations of the electrolyte. There is one circulation, which is designated the step circulation, in which the step circulating pump draws the electrolyte from the last cell of the step, sending the electrolyte to a more elevated tank (temperature tank)

EXAMPLE OF STEP ARRANGEMENT OF PLANT—A.
 Copper contents of the electrolyte of the steps held in geometrical ratio $\frac{1}{4}$.

Diagram shows solution circulation only.

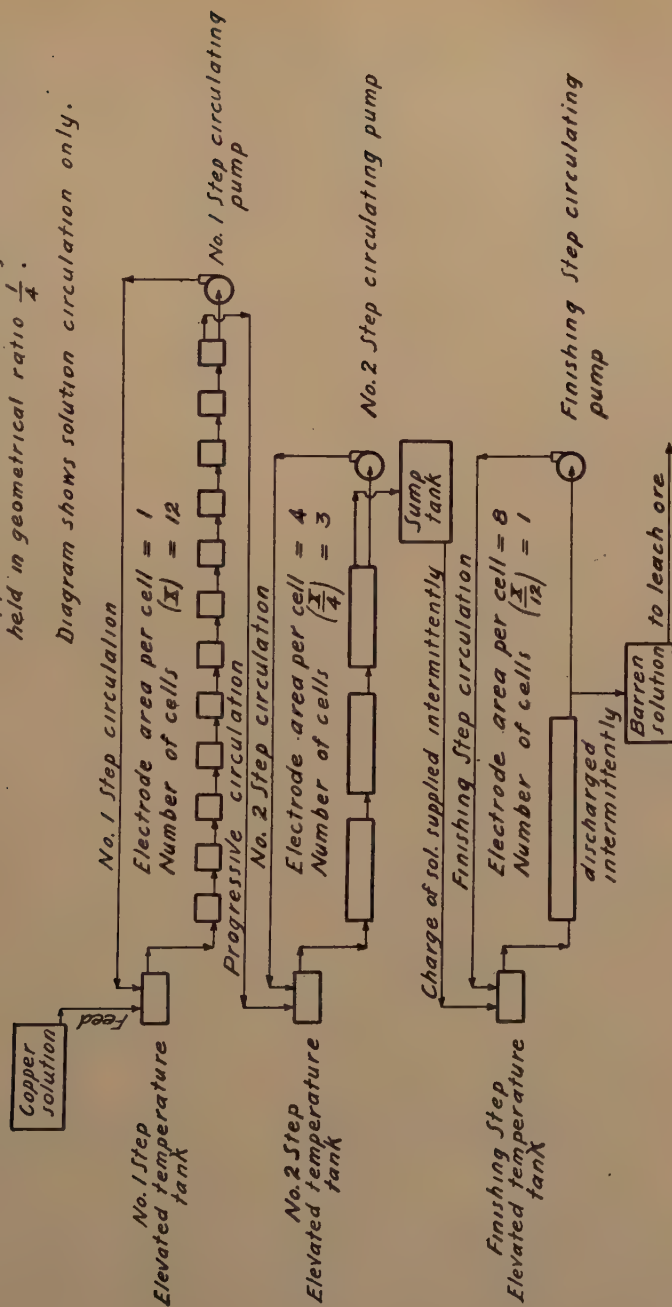


FIG. 16.—STEP ARRANGEMENT OF PLANT—A.

TABLE III.—*Example of Step Arrangement of Plant—Arrangement B*
Copper Contents of the Electrolyte of the Steps held in Geometrical Ratio $\frac{1}{2}$ (except the Finishing Step)

	Feed-Copper Solution	No. 1 Step	No. 2 Step	No. 3 Step	No. 4 Step	Finishing Step
Relative copper contents of electrolyte.....	1	$\frac{1}{2}$	$\frac{1}{4}$	$\frac{1}{8}$ -0
Copper contents of electrolyte, per cent.....	6.0	2.0	0.667	0.667-0
H ₂ SO ₄ contents of electrolyte, per cent.....		6.16	8.22	8.22-9.24
Copper remaining in electrolyte, expressed as per cent. of that in feed.....		33.3	11.1	0
Copper deposited, expressed as per cent. of that in feed.....		66.7	22.2	11.1
			(Nos. 1 and 3 Steps)			
Total copper deposited, expressed as per cent. of that in feed.....		66.7	88.9	(All Steps) 100
Number of cells in series in each step.....		1	$\frac{1}{2}$	$\frac{1}{4}$
Relative electrode area per cell.....		1	3	6
Relative C. D. in each step.....		1	$\frac{1}{2}$	$\frac{1}{4}$

in which it may be heated (maintaining the temperature of the system constant), and from which it overflows and gravitates into the first cell of the step. The electrolyte then passes on through the series of cells in the step and finally again to the same circulating pump.

There is the other circulation, which is ordinary progressive movement of the solution through the plant. The copper solution resulting from the leaching of the ore, or otherwise obtained, is run, together with the discharge of the No. 1 Step circulating pump, into the elevated temperature tank of the No. 1 Step. Thus more solution enters the first cell of the step than is drawn away by the circulating pump, consequently an equivalent amount of solution must leave the last cell of the step by way of the overflow discharge. This overflow discharge passes on and joins the discharge of the circulating pump of No. 2 Step, and enters the elevated temperature tank connected with that step. Finally the last cell of the step preceding the Finishing Step, overflows an amount of solution equal in amount to the inflowing copper solution fed to the elevated temperature tank of the No. 1 Step.

The copper contents of the solutions is maintained in geometrical ratio or in arithmetical difference from that of the copper solution fed to No. 1 Step to that of the electrolyte of the step preceding the Finishing Step.

The experimental plant was designed so that a variety of factors could be used. Let us select for a plant a solution resulting from the leaching of the ore with copper contents of 6.0 per cent. and the geometrical ratio $\frac{1}{4}$ for the copper contents of the electrolyte of the different steps. (See Table II and Fig. 16.) Then the electrolyte of No. 1 Step

EXAMPLE OF STEP ARRANGEMENT OF PLANT—B.
 Copper contents of the electrolyte of the steps held in geometrical ratio $\frac{1}{3}$.

Diagram shows solution circulation only.

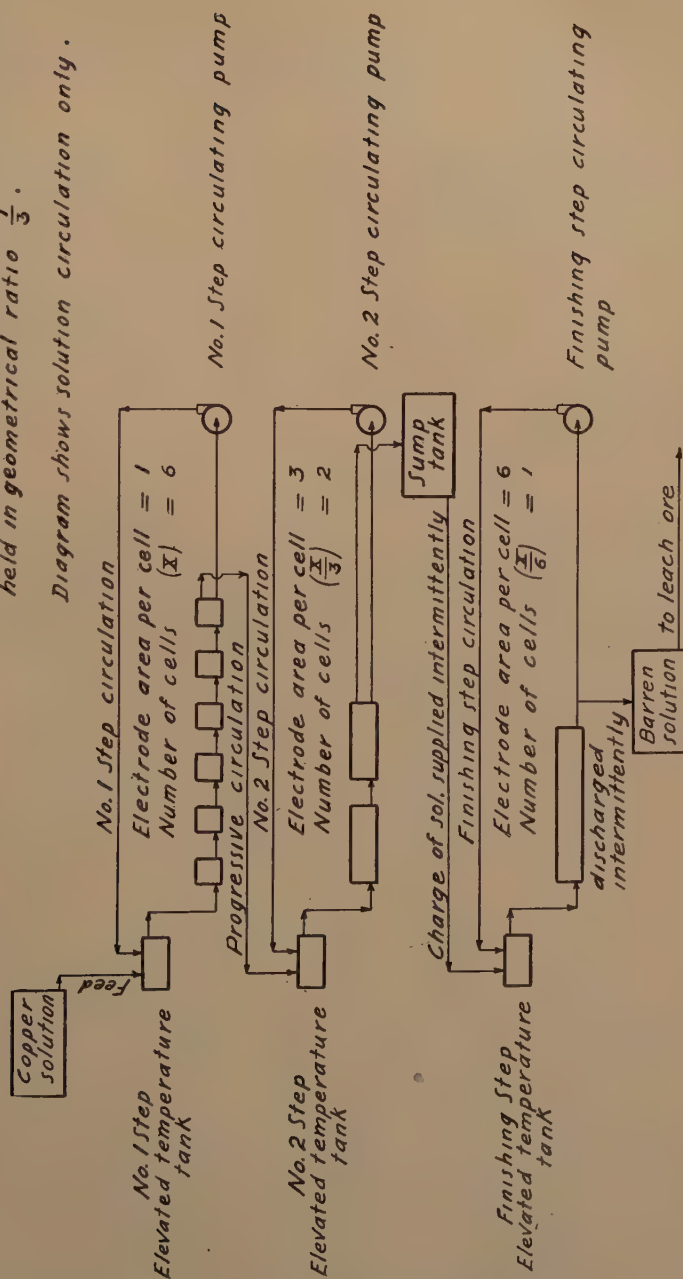


FIG. 17.—STEP ARRANGEMENT OF PLANT—B.

TABLE IV.—*Example of Step Arrangement of Plant—Arrangement C*
Copper Contents of the Electrolyte of the Steps held in Geometrical Ratio $\frac{1}{2}$ (except the Finishing Step)

	Feed-Copper Solution	No. 1 Step	No. 2 Step	No. 3 Step	No. 4 Step	Finishing Step
Relative copper contents of electrolyte....	1	$\frac{1}{2}$	$\frac{1}{4}$	$\frac{1}{8}$	$\frac{1}{16}$	$\frac{1}{16}$ —0
Copper contents of electrolyte, per cent....	6.0	3.0	1.5	0.75	0.375	0.375—0
H ₂ SO ₄ contents of electrolyte, per cent....		4.62	6.93	8.08	8.66	8.66—9.24
Copper remaining in electrolyte, expressed as per cent. of that in feed.....		50	25	12.5	6.25	0
Copper deposited, expressed as per cent. of that in feed.....		50	25 (Nos. 1 and 2 Steps)	12.5 (Nos. 1, 2 and 3 Steps)	6.25 (Nos. 1, 2, 3 and 4 Steps)	6.25 (All Steps)
Total copper deposited, expressed as per cent. of that in feed.....		50	75	87.5	93.75	100
Number of cells in series in each step....	x	x	2	4	8	8
Relative electrode area per cell.....	1	2	4	8	16	16
Relative C. D. in each step.....	1	$\frac{1}{2}$	$\frac{1}{4}$	$\frac{1}{8}$	$\frac{1}{16}$	$\frac{1}{16}$

will contain 6.0/4 equals 1.5 per cent. copper, and 6.93 per cent. H₂SO₄. Let us further assume for No. 1 Step 12 cells in series, each with one unit of area of cathode surface. Next select the most suitable current density for this step. The copper contents of the electrolyte is the governing factor in making this selection, since it is desirable always to maintain current density proportional to copper contents of the electrolyte. The selection of current density fixes the strength of current. The copper solution is fed to this step at such a rate that the copper deposited on the cathode amounts to three-quarters of the entering copper. The electrolyte circulating in the step thus remains constant in copper contents and in acid, and the solution overflows in volume equal to that of the inflowing feed solution.

The solution fed to No. 2 Step, being the overflow of No. 1 Step, contains 1.5 per cent. copper and 6.93 per cent. H₂SO₄. Since the electrolyte of this step contains one-quarter as much copper as that of the preceding step, then in order to maintain the electrolyte of this step constant one-quarter as much copper must be deposited on the cathodes of this step. This is accomplished by placing one-quarter as many cells in series, namely, using three cells. These cells should each have four units of cathode area, that is, a cathode area four times as great as that employed in each cell of No. 1 Step, since in this Step (No. 2) the electrolyte contains one-fourth as much copper, which requires a current density one-fourth that of No. 1 Step to be employed.

It is seen that the solution overflowing from No. 2 Step contains but 6.25 per cent. of the original copper contents. This solution in which the acid has been regenerated may pass on and be used to leach ore. That

it contains a small amount of copper is no detriment since this copper will return in the enriched solution and will not be lost.

In case it should be desired to extract the copper to the last trace, one more step—a finishing step—may be added to the plant. This step would be supplied intermittently with a charge of solution from the sump tank,

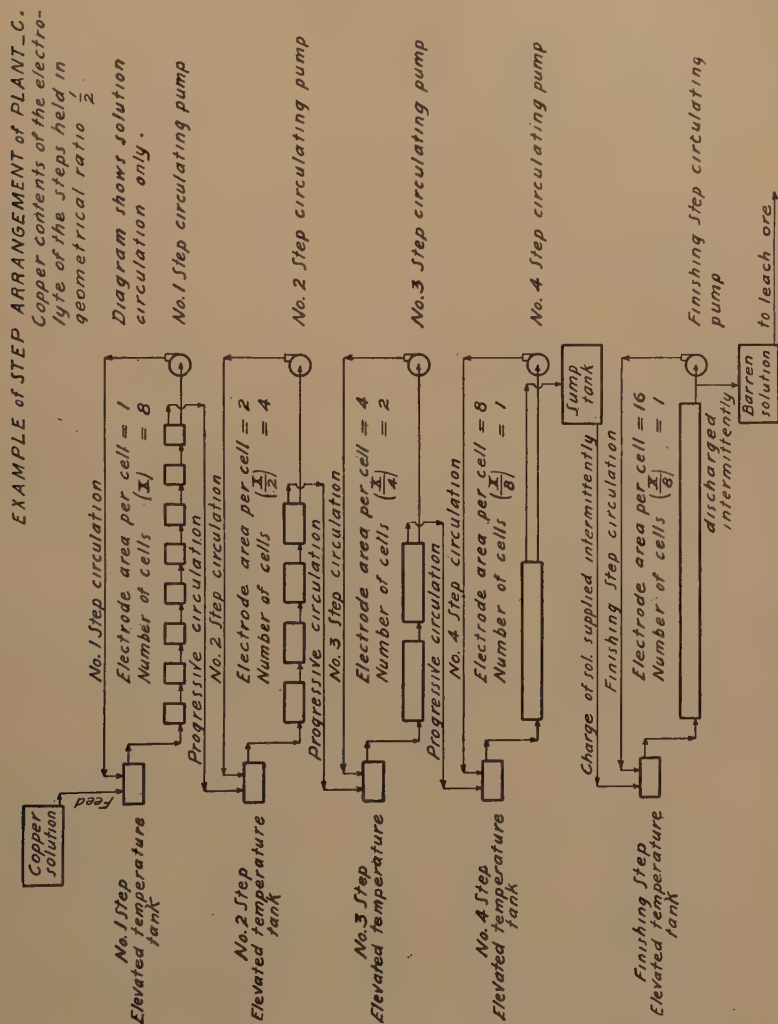


FIG. 18.—STEP ARRANGEMENT OF PLANT—C.

where the solution overflowing from No. 2 Step has collected. This charge of solution would be circulated in the Finishing Step by a circulating pump in the same manner as in the preceding step but with neither feed nor overflow, until all of the copper contents is deposited on the cathode. The solution, barren in copper, would be withdrawn from the system, after which a new charge would be supplied.

TABLE V.—*Example of Step Arrangement of Plant—Arrangement D*
Copper Contents of the Electrolyte of the Steps held in Arithmetical Difference, 1.5 Per Cent. (except the Finishing Step)

	Feed-Copper Solution	No. 1 Step	No. 2 Step	No. 3 Step	No. 4 Step	Finishing Step
Relative copper contents of electrolyte....
Copper contents of electrolyte, per cent....	6.0	4.5	3.0	1.5	1.5-0
H ₂ SO ₄ contents of electrolyte, per cent....	2.31	4.62	6.93	6.93-9.24
Copper remaining in electrolyte, expressed as per cent. of that in feed.....	75	50	25	0
Copper deposited, expressed as per cent. of that in feed.....	25	25	25	25
			(Nos. 1 and 2 Steps)	(Nos. 1, 2 and 3 Steps)		(All Steps)
Total copper deposited, expressed as per cent. of that in feed.....	25	50	75	100
Number of cells in series in each step.....	x	x	x	x
Relative electrode area per cell.....	1	1.5	3	6
Relative C. D. in each step.....	1	$\frac{2}{3}$	$\frac{1}{3}$	$\frac{1}{6}$

The number of cells in series in the Finishing Step would be one-third the number used in No. 2 Step. Then this step must operate continually to deposit all the copper sent to it by the overflow of No. 2 Step, since the amount of copper contained in the overflow solution of No. 2 Step is one-third the amount deposited in that step. The cells of the Finishing Step should each have (in this example there is but one cell) eight units of cathode area, that is, twice the cathode area and one-half the current density as that employed in each cell of No. 2 Step, as in this step (Finishing Step) the electrolyte has on an average one-half the copper strength.

Even when the Finishing Step is employed, it may be desirable to stop the electrolysis somewhat short of complete extraction. (This matter is discussed in connection with Tests Nos. 2 and 4.)

In order to obtain the figures given above, it is necessary to compensate for evaporation, which at the temperature employed (50° C.) and with the large surface of solution exposed is considerable. In Test No. 1, evaporation was compensated for by the addition of water to the temperature tank of each step, equal in amount to that of the evaporation. Thereby was demonstrated the feasibility of maintaining the composition of the electrolyte constant, both in copper and acid, during the electrolysis, and with copper contents of the electrolyte of the different steps in the above-described ratios (4:1)

In Tests Nos. 2 and 4 the plan of operation was slightly modified from that followed out in Test No. 1 in order to somewhat simplify the operations. To the temperature tank of No. 1 Step was fed a more dilute copper solution, which amounted to the original volume of the 6.0 per cent. copper solution plus the required amount of water to compensate for evaporation of No. 1 Step. Consequently No. 1 Step oper-

EXAMPLE of STEP ARRANGEMENT of PLANT — D.
Copper contents of the electrolyte of the steps held in arithmetrical difference, 15 percent.

Diagram shows solution circulation only.

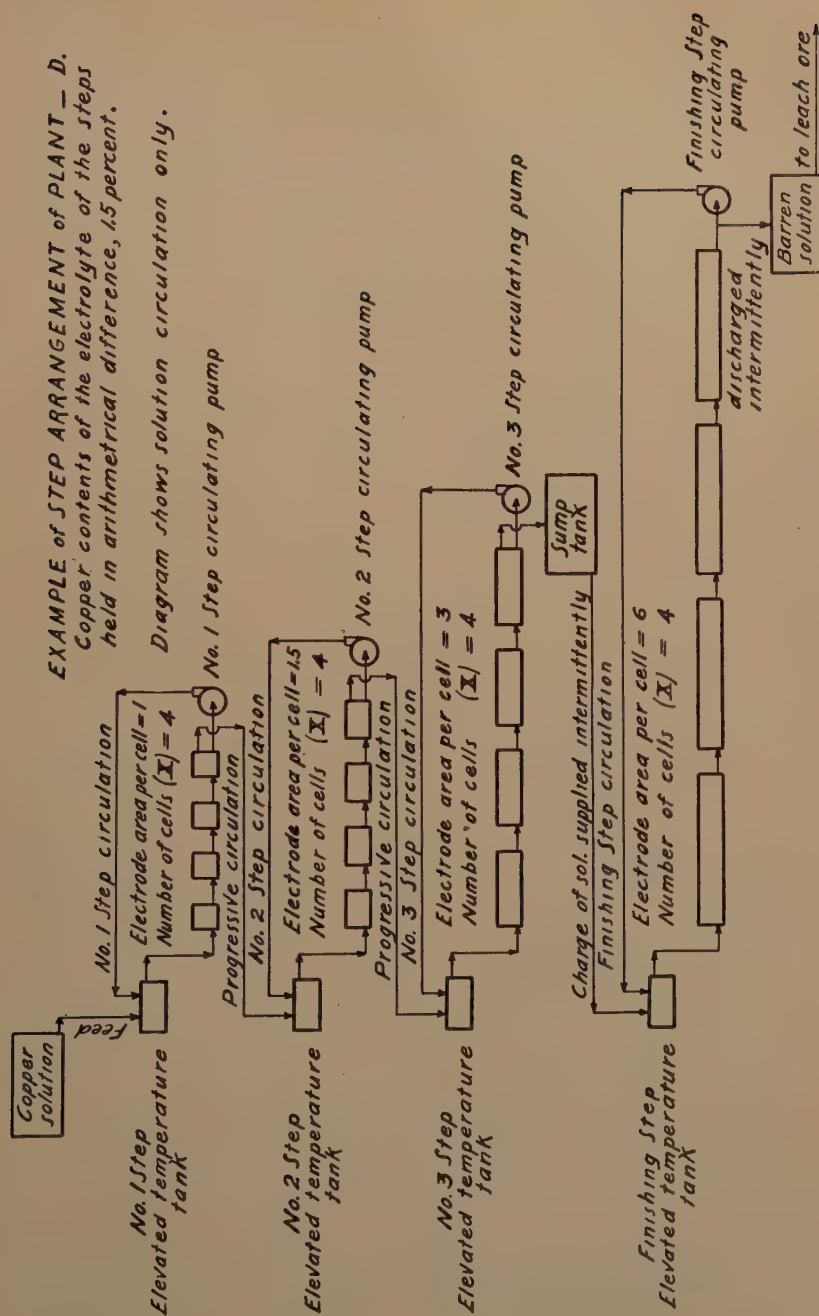


FIG. 19.—STEP ARRANGEMENT OF PLANT—D.

ated in every way exactly as it did in Test No. 1, thus maintaining the composition of its electrolyte the same and causing an overflow equal in amount and composition to that in Test No. 1. This calling for a more dilute copper feed solution might be an advantage in practice, as it might be easier to secure than the one of higher copper contents. Further dilution of the copper solution fed to No. 1 Step, to compensate for evaporation in No. 2 Step, is not permissible as it would derange the constant conditions desirable to be maintained in that step. Moreover, for simplicity of operation, water was not added to No. 2 Step to compensate for evaporation, consequently the copper contents of the electrolyte of No. 2 Step differed from the original geometrical ratio. A higher value of both the copper and the acid contents obtained. Such higher values of both copper and acid, however, remained constant. The higher copper value would permit of a somewhat smaller electrode surface and higher current density per cell than was called for in Test No. 1.

The ideal of the copper hydro-metallurgist is to secure, in the hydro-electrolytic extraction of copper from its ores with insoluble anodes, conditions comparable with those which obtain in the electrolytic refining of copper with soluble anode. By means of the step system it is believed that these conditions are more nearly approached than they have been heretofore. The desirable conditions are: (a) Constant composition of electrolyte with current density adjusted to suit composition; (b) small power consumption.

The step system arrangement of the electrolytic cells accomplishes "a" and also permits the electrolyte to be circulated at any rate desired as in electrolytic refining of copper. A rapid rate of circulation in the individual cells which is made possible by the step system, together with the adjusting of the current density proportionate to the copper contents (which is held constant in each step), makes possible the securing of high current efficiency.

As most copper ores contain sulphides, they should be roasted prior to leaching. Sulphur dioxide gas is evolved. In such cases it may be desirable to pass this gas through the electrolytic cells, utilizing it as a depolarizer and at the same time producing additional sulphuric acid for the leaching. Ordinarily in leaching copper ores, the solution takes up some iron from the ore. Were no sulphur dioxide gas introduced into the cell, the ferrous sulphate would become oxidized to ferric sulphate at the anode. The ferric sulphate formed would be carried by circulation of the electrolyte to the cathode where it would dissolve some of the deposited copper, again becoming ferrous sulphate, after which the cycle would be repeated. Current efficiency would thus be decreased. But the sulphur dioxide gas when employed with a suitable insoluble anode, besides maintaining a high current efficiency, is beneficial in reducing the power

consumption to a point more comparable with that consumed in the electrolytic refining of copper with soluble anode. (See Figs. 14 and 15.)

Increased areas of electrodes to give reduced current density for corresponding depletion in copper contents of the solution in the process of electrolysis have been used heretofore. In these processes, however, there is but one circulation of the electrolyte, namely, that through the plant progressively; the solution becomes depleted of the copper during its passage. Such a circulation would necessarily be slow and insufficient for securing the best results. Moreover, the electrolyte, due to the slow progress through the cells, would vary in composition in different parts. So, although in the ordinary processes the aim is to maintain the current density proportional to the copper contents, it has not really been accomplished.

The step arrangement of the plant is such that—

1. Rapid circulation is maintained by the step circulating system.
2. The composition of electrolyte remains absolutely constant in each step of the plant.
3. The current density is held strictly proportional to the copper contents of the electrolyte.

SUMMARY

It was desired to select an Arizona problem, so the treatment of a porphyry copper ore seemed to be one of the most important. This kind of ore when treated for the extraction of its copper by mechanical concentration and smelting, the methods most generally employed, yields 66 per cent. or less of its copper.

In these experiments, therefore, the aim was to determine a better method of treatment. The method selected for investigation was one in which metallurgists are at present doing much experimenting in the hope of demonstrating the superiority of leaching and electrolytic precipitation over earlier practice, namely: 1, Oxidizing roast; 2, leaching with dilute sulphuric acid; 3, electrolytic precipitation of the dissolved copper.

While it was planned to carry on a complete systematic test of the ore, thereby enabling one to determine the suitability of this ore to the hydro-electrolytic treatment, special stress was laid on roasting and on electrolytic precipitation using sulphur dioxide to lessen the consumption of power and to produce additional sulphuric acid.

It has been demonstrated in these experiments that it is feasible successfully to roast the ore, so that when leached with hot dilute sulphuric acid, the entire copper contents may be obtained in solution.

Some of the practical engineers say that they secure but little beneficial reduction in power consumption when using sulphur dioxide gas as a

depolarizer. Tossizza says, "I have thought to use . . . insoluble anodes kept in contact with sulphurous acid, and thus to utilize the known depolarization properties of the said sulphurous acid. These anodes can be made of carbon, and in this case the sulphurous acid can be introduced outside the anode or in the interior thereof;" "One can thus obtain a very beautiful deposit of pure electrolytic copper . . . with a sufficient intensity at a voltage of about six-tenths of a volt." The experiments reported under Tests Nos. 3 and 4 corroborate Tossizza's statement in that the method of introducing the sulphur dioxide into the electrolyte is unimportant, provided only that it be introduced in sufficient quantity to keep the electrolyte saturated.

Regarding the saving of power (depolarization), these experiments show depolarization by the use of sulphur dioxide only when used in connection with carbon anodes, while Tossizza, although specifically suggesting a carbon anode, intimates that the same may be obtained with other insoluble anodes. Tossizza does not state the current density at which he was operating when depositing copper with the extremely low voltage of six-tenths of a volt, although "with a sufficient intensity" might mean commercial current density. Experiment Test No. 3 shows likewise that copper deposits continuously at six-tenths of a volt when current density equals 5.8 amperes per square foot, depositing 4 lb. of copper per kilowatt-hour. (See Fig. 14.) It is to be noted that although beneficial depolarization and consequent saving of power is secured at all current densities, when SO_2 is introduced into the electrolyte, it is relatively not the same in amount but decreases as the current density increases.

It is to be hoped that future experiments will demonstrate how to secure the same beneficial depolarization with sulphur dioxide when electrolyzing with high current densities. Likewise whether, and how, beneficial depolarization by sulphur dioxide may be secured with other kinds and types of insoluble anodes.

When it is desired to extract the copper from the electrolyte down to a small trace, sulphur dioxide is very beneficial, not only in reducing the power consumption but in causing the copper to deposit more firmly on the cathode, which when sulphur dioxide gas is not used forms as a spongy deposit toward the last of the electrolysis. On attempting to extract the last trace of copper from the electrolyte, sulphide of copper forms on the cathode.

Regarding the lead anodes, there is no depolarization when used with or without sulphur dioxide gas. There was no loss in their weight during the test.

When starting out on this line of investigation, it was found necessary to design and construct a complete plant on a miniature scale. In doing this, a novel idea appeared—the step arrangement—which greatly facili-

tated the work in all stages of the process, especially in the finishing step where the copper was extracted down to a trace with the production of a good firm cathode, when sulphur dioxide gas was used.

Test No. 1 demonstrates, when using the step system, the feasibility of depositing copper continuously from the electrolyte, while at the same time the copper contents of said electrolyte remains undiminished. A plant composed of several steps of such cells may be operated so that a liquor strong in copper flows into the upper step of the plant continuously from the leaching vats at such a rate that a liquor depleted of its copper, in which the acid solvent has been regenerated, outflows from the plant continuously to the leaching plant. The advantages of a step arrangement of plant are apparent in that the electrolysis is conducted under constant invariable conditions in each cell of the respective steps, as well as in every part of the cell. The cathode area of a step may easily be adjusted so that the most desirable current density is secured (as determined by experience in operating). It is to be noted in step arrangement of plant, the electrolyte may be circulated in any of the steps at any desired rate. This has been shown to be beneficial in promoting high current efficiency.

The experiments demonstrate that the porphyry ore, when treated by a hydro-electrolytic process, will yield its entire copper contents as a good grade of cathode copper. It is therefore hoped that by using a hydro-electrolytic method, lower-grade copper porphyry deposits may be worked than formerly could be worked by methods of concentration followed by smelting.

Epitome

1. The most suitable temperature for roasting the Arizona porphyry copper ore, containing sulphides, in order to render it amenable to acid leaching methods, is between 600° C. and 725° C. The more finely ground the material the shorter the time required for the roasting so as to produce the maximum amount of soluble copper: materials which will pass through a 20-mesh screen and remain on 80 mesh require about 2 hr. roasting at 600° C. to 725° C.; and when ground to pass through an 80-mesh screen the time required is about 1¼ hr. If the roasting is concluded at temperatures above 800° C., the oxidized copper is converted into a compound which is insoluble in dilute sulphuric acid. The longer the roasting is conducted above 800° C., the greater the amount of insoluble copper produced.

2. A heated solution is necessary to leach efficiently the copper from the roasted material. A 10 per cent. H_2SO_4 solution at 100° C. leached out, in from 3 to 6 hr., all the copper from all roasted materials (through 40 on 80 mesh, through 80 mesh, whole through 20 mesh), except material through 20 on 40 mesh, in which case the extraction was not so high.

3. The nature of the anode is an important factor in securing depolarization by sulphur dioxide gas:

4. Depolarization with consequent saving in power is accomplished when using sulphur dioxide gas with a carbon anode, while there is no depolarization with a lead anode.

5. The depolarization by sulphur dioxide gas, even with carbon anodes, does not reach the theoretical amount, being between 45 and 65 per cent.

6. The amount of depolarization effected by sulphur dioxide gas when used with carbon anodes varies with the current density, being a maximum at low current density.

7. The method of introducing the sulphur dioxide gas into the cell is unimportant. All that is necessary is that it be introduced in some way, and in such quantity that the electrolyte is saturated with the gas, so that there is some escaping by bubbling at the surface.

8. A smoother deposit of copper forms when using sulphur dioxide gas as a depolarizer than when not using it.

9. In the electrolysis of an acid solution of copper sulphate, when sulphur dioxide is not supplied to the cell and when one is endeavoring to carry electrolysis to the point of complete extraction, a soft spongy deposit begins to form on the cathode before the complete extraction of the copper is effected. There is also a considerable rise in polarization when the copper contents of the electrolyte becomes low.

10. In the electrolysis of an acid solution of copper sulphate, when sulphur dioxide is supplied to the cell, the copper contents of the electrolyte may be reduced to a very small trace with the formation of a good, firm cathode, without rise in polarization toward the end. Current density and energy efficiency remain high to the end. It is only when prolonging the operation beyond the time when but a small trace of copper remains that sulphide of copper forms as a thin coating on the cathode.

11. Lead anodes do not peroxidize or deteriorate appreciably when used with or without the introduction of sulphur dioxide into the electrolyte.

12. A novel idea—step arrangement of process—makes feasible the depositing of copper continuously, while the copper contents of said electrolyte in the respective steps remains constant. A plant composed of several steps of such cells may be operated so that a liquor strong in copper flows to the plant continuously, and the liquor which outflows from the plant is depleted of its copper.

13. The circulation of the electrolyte by the step arrangement increases the current efficiency. Such rapid circulation is not possible in plants as ordinarily arranged.

Electro-Metallurgical Industries as Possible Consumers of Electric Power

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(San Francisco Meeting, September, 1915)

I. INTRODUCTION

THE utilization of hydro-electric power in electro-metallurgical industries, aside from purely mechanical operations, may be of two kinds. The electric energy may be used to supply the heat necessary for the performance of the chemical reactions of a metallurgical process, or the electrolytic action of a direct current may be employed for extraction of metals, either in a reduction process or a refining process. In discussing this subject, it is well to distinguish between established industries, which are producing metals and alloys at a profit by use of the electric current, and the many processes which are either still in the experimental stage or have not even passed the paper stage of development of a patent. That is, we consider electro-metallurgical industries as commercial operations which are now consuming large blocks of electric energy, while processes which are not in commercial operation are only prospective consumers. Also, owing to the difficulty and risk of any attempt to foresee the effect of the European war upon electro-metallurgical industries, we have felt obliged to disregard the war entirely, and state our opinions regardless of its existence.

Application of Electric Energy to Metallurgy

Disregarding electrolytic refining processes, the applications of electric energy to metallurgy, when considered from a commercial viewpoint, may be classified as follows:

Industries	Experimental Processes
Manufacture of aluminum	
Electric-furnace manufacture of:	Electric-furnace smelting of:
Ferro-alloys	Copper ore
Pig iron	Zinc ore
Steel	

In this paper only the above industries and experimental processes will be considered as possible consumers of hydro-electric power in large quan-

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tities. Electrolytic refining processes are not included above, since they are either so closely interwoven into a cycle of metallurgical operations as one single step as to prohibit their operation as separate enterprises, or the electrolytic process is on such a small scale of operation as to be unimportant as a consumer of power. An example of the former is the electrolytic refining of copper. As at present operated, the electrolytic copper refinery, while it may be located at the copper smelter, is usually owned by a smelting company upon which it largely depends for its supply of unrefined anode copper. It has also been found advantageous to locate copper refineries near the market for refined copper, which is New York. An extreme illustration of a small-scale electrolytic operation which is not worth considering as a large consumer of power is the electrolytic refining of gold and silver.

All of the above electro-metallurgical industries are adaptable, under proper conditions, to self-contained development in a single plant. Practically none of these industries consume less than 1,000 hp. in a single installation. There are many electro-metallurgical processes in successful use for the extraction or refining of metals which do not consume large amounts of power, because the yearly yield is small, as in the case of gold and silver; or production is reduced by the small market, as, for example, for calcium, magnesium, or bismuth. If there were a demand for them, the two former metals might be produced profitably in conjunction with the manufacture of sodium, which, while a larger consumer of power than either calcium or magnesium, is not a large consumer in comparison with other electro-metallurgical industries.

There are many electro-metallurgical processes which might be considered as possible consumers of power, but in the majority of cases they exist only as patent specifications and have not yet been tested by either small or large experimental work. As an immediate market for power, most of these processes are of no interest to the power producer.

Factors Influencing the Success of an Electro-metallurgical Industry

The commercial success of an electric furnace or electrolytic industry, in which the basic process employed is technically sound, may depend largely upon the following factors: (1) the market for the product; (2) the cost of electric power; (3) freight rates; (4) labor conditions; (5) cost of raw materials.

Not only the cost of electric power, but all the other factors named, will influence, to a considerable degree, the success of any manufacturing project. If there is not a steady market for the product, permitting a profit on the investment, a low cost of power will not insure business success. There is but a limited market for many electro-metallurgical products, so that while a good profit might be shown on paper by comparing the cost of production with the average selling price, the enterprise

would be a failure with a large part of its products unsold. In the face of strong competition, success depends to a large extent, as in any other line of business, upon the efficiency of operation, quality of the product, and ability to meet competitive prices.

Reduction processes, as a general rule, require a very low power cost, especially those operations producing a large tonnage of a comparatively cheap product, which compete with combustion processes, as, for example, the electric smelting of iron ore and zinc ore. On the other hand, electric-furnace refining processes do not require for commercial success an extremely low power cost. Many electric steel furnaces are operated at a profit on a power cost of 1c. per kilowatt-hour, or \$65.70 per horsepower-year, while few electric-furnace reduction processes can operate profitably with a power cost of over 0.3c. per kilowatt-hour, or \$20 per horsepower-year, and for complete assurance of commercial success, the power cost should be as small as from \$10 to \$20 per horsepower-year.

But, we repeat, the commercial success of an electro-metallurgical enterprise does not depend entirely upon the cost of power. Freight rates have a large influence on it. Generally speaking, in the western part of the United States, no such enterprise producing a large tonnage would have much chance of success unless located within a few hundred miles of the sea-coast, because of the high freight rates prevailing in the West as compared with Eastern rates. The short distance to water shipment has been a large factor in the success of Norwegian and Swedish plants, as well as of those in Switzerland and the French Alps. Practically all of their product is exported to foreign countries by water, while most of their ore and coal or coke is shipped to them by sea. With the exception of some specific raw material near by, the cost of raw material will depend largely upon freight rates, for in the majority of cases, at least the ores used must be brought from a distance.

II. PRESENT COMMERCIAL STATUS OF ELECTRO-METALLURGY

Aluminum

All aluminum manufactured to-day is produced by an electro-metallurgical process, either by the Heroult process or the Hall process, which are essentially the same. In the United States the Hall process is used and in Europe the Heroult process.

The production of aluminum in the United States has not increased materially since 1912 but exact figures are difficult to obtain, because the available statistics for recent years are those of consumption. The consumption in 1913 amounted to 72,379,090 lb., of which probably about 30,000,000 lb. was imported during the calendar year. Importations during the year ending June 30, 1913, amounted to 26,642,112 lb., and imports were increasing at the end of that period, but during the last quarter

of 1913 the consumption of aluminum decreased largely, in common with the reduction of business activity in other lines throughout the United States. The actual production of 1913 in the United States may be roughly estimated at 42,000,000 lb. In 1907, 60,000 hp. was being utilized in the manufacture of aluminum in the United States. To-day probably 80,000 hp. is in use for this purpose.

Until recently the production of aluminum in the United States was entirely controlled by the Aluminum Co. of America, owners of the Hall patents, which have now expired. In this country that company has one plant in operation at Niagara Falls, N. Y., and one at Massena, N. Y.; and in 1914 it started a new plant at Maryville, Tenn., where it is reported that 100,000 hp. can be developed. In Canada, it has a plant at Shawinegan Falls.

On the expiration of the Hall patents in the United States, a company was formed by French financiers interested in the Aluminium Française, under the name of the Southern Aluminium Co. A plant is in course of construction at Whitney, N. C., on the Yadkin River, where a total development in high water of 100,000 hp. is possible, although the average power will be about 45,000 hp. The capacity of the plant to be erected will be 20,000,000 lb. of aluminum per annum. This company continued its construction work up to the outbreak of the European war, when it became embarrassed financially, and was obliged to discontinue its work. The Aluminum Co. of America is still, therefore, the only producer of new metal in the United States, the remaining production being furnished by concerns that resmelt scrap and junk.

In Europe the aluminum industry has progressed as rapidly as in the United States. From the table given below it will be seen that Europe produced in 1913 about three times as much aluminum as the United States; but this country leads in individual production, with France second.

TABLE I.—*World's Production of Aluminum*
(From Statistical Report of the Metallgesellschaft, Frankfurt-am-Main)

	1911 Metric Tons	1912 Metric Tons	1913 Metric Tons
United States.....	18,000	19,500	22,500
Canada (exports).....	2,300	8,300	5,900
Germany.....			
Austria-Hungary.....	8,000	12,000	12,000
Switzerland.....			
France.....	10,000	13,000	18,000
England.....	5,000	7,500	7,500
Italy.....	800	800	800
Norway.....	900	1,500	1,500
Total	45,000	62,600	68,200

Ferro-Alloys

The growth of the ferro-alloy industry in Europe has been rapid since 1899, but comparatively slow in the United States. There are about 25 European plants engaged in the manufacture of ferro-alloys by the electric-furnace method, as compared with two in the United States. There is, however, an electric-furnace ferro-silicon plant in Canada, at Welland, Ont.

There are several reasons why the growth of this industry has been slower in America than in Europe. Hydro-electric power is not so cheap here, and not so favorably located for the receipt of raw material and the sale of product. The water-power sites cannot be developed as cheaply as many of the foreign sites, where the cost of electric power per horsepower-year varies from \$7 to \$15 as compared with \$15 to \$30 in the United States, for power delivered at the manufacturing-plant transformers. In Canada, power is somewhat cheaper; but is often located in inaccessible places. Most of the Norwegian and Swedish plants are located at tidewater, or on navigable rivers. French works are within a couple of hundred miles of Marseilles. The use of ferro-alloys in the manufacture of high-class steels did not advance as rapidly in the United States as in Europe, and owing to less favorable natural conditions, electro-chemical and electro-metallurgical industries in general have not had so rapid a growth here.

A large proportion of the ferro-alloys used in the United States are imported, since, although there is a duty, local manufacturers do not supply the whole demand. This is true of about one-half of the ferro-manganese and one-half of the ferro-silicon used in the United States, as well as a large part of the ferro-tungsten. More ferro-titanium and ferro-vanadium are manufactured here than abroad. Our ferro-chrome production just about supplies the local demand.

In the United States, the Titanium Alloys Manufacturing Co. has a plant for the production of ferro-titanium at Niagara Falls, N. Y. The Electrometallurgical Co. has a plant at Kanawha Falls, W. Va., and another at Niagara Falls, N. Y. This company makes ferro-silicon, ferro-chrome, ferro-tungsten, ferro-vanadium, ferro-molybdenum, and ferro-phosphorus. The Primos Chemical Co., Primos, Pa., manufactures metals and ferro-alloys by chemical methods or in combustion furnaces. Among its products are ferro-tungsten, ferro-vanadium, tungsten metal, ferro-molybdenum, ferro-chrome, ferro-nickel, and ferro-boron. The American Vanadium Co., Bridgewater, Pa., manufactures ferro-vanadium by a method similar to the thermit process. The Goldschmidt Thermit Co. has a plant for the manufacture of metals and ferro-alloys by the thermit process at Newark, N. Y., but imports most of its products from its foreign works. This company produces ferro-titanium, ferro-

TABLE II.—*Importations of Ferro-Alloys into the United States during the Year Ending June 30, 1913*

	Rate of Duty	Quantity Tons	Value	Value per Unit
Ferro-manganese	\$2.50 per ton	128,136.55	\$5,484,829	\$42.80
Ferro-silicon, over 15 per cent. silicon.....	20 per cent.	9,257.18	574,494	62.06
Chromium and ferro-chrome, value \$200 per ton or less....	25 per cent.	459.09	53,624	116.80
Value more than \$200 per ton.....	20 per cent.	68.29	35,667	63.55
Ferro-phosphorus, value \$200 per ton or less.....	25 per cent.	81.38	3,689	45.33
Molybdenum and ferro-molyb- denum, value more than \$200 per ton.....	20 per cent.	7.00	15,939	3,187.80
Titanium and ferro-titanium, value \$200 per ton or less....	25 per cent.	7.00	306	43.71
Value more than \$200 per ton.....	20 per cent.	19.21	9,213	479.59
Tungsten and ferro-tungsten, value more than \$200 per ton	20 per cent.	654.30	795,467	1,215.75

vanadium, ferro-molybdenum, ferro-silicon, ferro-chrome, and chromium, as well as other metals and alloys. It has been recently reported that the Noble Electric Steel Co., Heroult, Cal., is manufacturing ferro-manganese at its electric-furnace plant.

Imports of ferro-alloys into the United States during the year ending June 30, 1913, are given in the table above. The rates of duty were those in force previous to the passage of the present tariff, under which all duties were reduced considerably, in most cases down to 15 per cent. ad valorem. Hence it may reasonably be expected that imports will increase at some time in the future.

Pig Iron

The electric furnace for smelting iron ore is of advantage only in localities where charcoal and coke are expensive, and electric power is cheap. In the manufacture of pig iron, the electric furnace consumes one-third of the carbon used by the blast furnace, and hence its use may be advantageous where coking coal is scarce, and charcoal expensive. In considering the electric smelting of iron ores with regard to its commercial status at the present time, it must be remembered that by reason of the cheapness of water haulage, the electric furnace is in this case competing directly with the blast-furnace product, regardless of its location. The situation is not like that of the aluminum or ferro-alloy industry,

in which the electric furnace has the field to itself, because of its technical and commercial superiority over any combustion process. Hence, in proportion to the amount of pig iron produced, we cannot expect to show nearly as large a rate of increase for the electric-furnace process as in the case of aluminum and ferro-alloys.

While the electric furnace has been successful in the smelting of iron ores in other countries and in those districts in this country which are favorable to the same, there is only one electric furnace pig iron plant of two furnaces with a total capacity of about 7,000 hp. This plant is located at Heroult, California. Although the electric furnace uses only one-third of the coke or charcoal that the blast furnace uses, all solid reducing agents are so scarce and expensive on the Pacific Coast of the United States as to greatly increase the cost of electric smelting and practically prohibit the blast furnace smelting of iron ores. Attempts have been made to use oil, but these have not yet proved successful.

While advances in electric smelting of iron ore have been satisfactory, considering that its field of use is limited, the actual tonnage capacity of electric pig iron furnace is small. This is quite clear when it is remembered that the total erected capacity in power consumption of the electric iron-smelting furnaces (47,000 hp.) would produce about the same amount of pig iron per day as one modern blast furnace of 450 tons output per 24 hours.

Steel

As in other electro-thermic processes, development has not been so rapid in the United States as in Europe. Only 14 furnaces are in this country. The average capacity per charge of the furnaces already built is 3.7 tons, whereas that of the furnaces under construction is 4.5 tons, an increase of 21.6 per cent. The total charge capacity of the furnaces now installed is about 250 tons per charge, and the total charge capacity of the furnaces under construction will be 170 tons per charge. The arc furnaces vary in capacity from 1 to 15 tons and require from 200 to 1,500 kw. for operation.

Table III gives the annual production of steel in electric furnaces, by countries, for the years 1908 to the first half of 1912.

In the first years of its development, the electric-furnace process was considered as a competitor of the crucible process only, for making high-class steel from scrap iron and scrap steel; but with the successful operation of larger furnaces the electric process is likely to become an important adjunct to the Bessemer and open-hearth processes as a means of super-refining their molten products. The electric process, however, does not appear to be destined to supersede either of these methods, since greater efficiency and economy are obtained by a combination of any two of the three processes as a duplex process. The success of recent experiments

TABLE III.—Yearly Production of Electric-Furnace Steel

Country	1908			1909			1910			1911			1912
	Production	Number of Furnaces	Tons	Production	Number of Furnaces	Change in Production	Production	Number of Furnaces	Change in Production	Production	Number of Furnaces	Change in Production	First Half
Germany and Luxemburg...	19,536	8	17,773	8	8	9.0	36,188	13	+104.1	60,654	15	+ 67.8	Tons
United States.....	55	1	13,762	4	4	52,141	7	+280.0	29,105	9	- 44.2	6,882
Austria-Hungary.....	4,333	9,048	109.2	20,028	+120.8	22,867	10	- 14.0
France.....	2,686	7	6,515	12	12	143.0	13,445	21	+106.2	13,850	21	- 3.0	7,920
Sweden.....	591	11	11	431	12	- 27.0	2,034	13	+372.0
Norway.....	1	1
England.....	12
Italy.....	5	5	2	4
Switzerland.....	2
Belgium.....	2
Russia.....	1
Total.....	26,610	16	47,039	40	40	78.2	120,116	56	+155.4	126,476	90	+ 5.22

has obtained for the electric process a definite place as a super-refining method. In time, preliminary refining will probably be done mainly in the Bessemer converter, the process being finished in the electric furnace, or the open hearth. In Europe, the electric-furnace process for making steel of the highest grades is rapidly superseding the old crucible method, because of its greater economy of operation and the possibility of using materials of lower grade.

Copper

So far as we are aware, no copper ores are treated in the electric furnace in this country at the present time.

However, more or less experimental work has been done upon the subject, and, as a result of this work and reasoning by analogy, there seems no good reason why copper-bearing ores cannot be as successfully treated in an electric furnace as in a combustion furnace.

Zinc

Although, as previously remarked, more progress has been made hitherto in the electric smelting of zinc ores than in that of any of the non-ferrous metals except aluminum, and metals forming ferro-alloys, such as silicon, chromium, and tungsten, the process is nevertheless still largely in the experimental stage. There is no plant operating on a commercial scale in this country. One small commercial plant is in course of erection at Keokuk, Iowa, by the Johnson Electric Smelting Co. It appears that the experiments conducted at Hartford, Conn., for several years have proved successful enough to warrant the installation of a small commercial unit to test the process further. The Johnson process and the Trollhättan process are essentially the same. Johnson claims to have overcome the problem of condensation of zinc vapor into zinc, instead of blue powder, which has been the great problem at Trollhätta.

III. THE POSSIBILITY OF DEVELOPING ELECTRO-METALLURGICAL INDUSTRIES IN THE WESTERN PART OF THE UNITED STATES

Having thus briefly stated the present status of the electro-metallurgical industries we can now consider what the chances would be for the development of such industries in the intermountain and Pacific Coast region of the United States. For the purpose of this paper, we will consider the manufacturing of the following electro-metallurgical products.

Aluminum

Raw Materials and Labor.—The ores and other raw materials necessary to the manufacture of aluminum are bauxite, cryolite, coal, and caustic.

Such being the case, all the raw materials necessary, with the exception of coal, would, at the present time at least, have to be brought in from some other point, if an attempt were made to manufacture aluminum in any of the intermountain and Pacific Coast States; and in California, it would be necessary to import coal as well. Bauxite is mined in the United States in Alabama, Arkansas, Georgia, and Tennessee. Arkansas is the leading producer. An aluminum plant located at any point in the western part of the United States would be obliged to depend for its source of bauxite upon some of these Eastern States, as bauxite is not known to exist in the Western United States in large quantities. It has been proposed by some that alunite might become a source of aluminum, but all processes for the extraction of aluminum from silicates are still very much in the experimental stage. About 10 per cent. of the bauxite consumed in the United States is imported from Europe. The Southern Aluminium Co., with its plant in North Carolina, proposed to import all of its bauxite from France during the first few years of operation.

The average price of bauxite at the mines, in 1913, was \$4.75 per long ton. A typical bauxite analysis follows:

	Per Cent.
Insoluble.....	12.13
Loss on ignition.....	28.97
Alumina (Al_2O_3)	57.56
Iron oxide (Fe_2O_3).....	1.34

About 4 long tons of bauxite are necessary per ton of aluminum, and will cost at the mine in Arkansas or Georgia, \$19. This bauxite, if used in the western part of the United States, must be hauled from the East by rail, or by rail and water. It is doubtful if raw bauxite could be laid down at any point in the West for less than \$14 per ton. Therefore the total cost of bauxite per ton of aluminum produced from it would be \$56. Four tons of bauxite would produce 2 tons of alumina. The cost of purification would bring the cost of alumina to \$38 per ton of 2,000 lb. Two tons of alumina produce 1 ton of aluminum, so that the cost per ton of aluminum with bauxite at \$14 would be \$76 per ton.

Freight could be saved by purification of bauxite to alumina at the mine or nearby. The cost of alumina at the mine would be about \$19.50 per short ton, or about \$28 to \$30 in the West. The alumina for 1 ton of aluminum would thus cost from \$55 to \$60, a saving of about \$9 per ton of alumina, or \$18 per ton of aluminum made from the alumina.

In the manufacture of aluminum, 0.1 ton of cryolite (sodium aluminum fluoride, AlF_3 , 3NaF), is consumed by volatilization of the electrolyte per ton of aluminum made. Cryolite is found only in Iceland. Some aluminum manufacturers substitute an artificial fluoride for the cryolite, using also some calcium fluoride, fluorspar. Cryolite is worth about \$24 per long ton, or 1.1c. per pound, in New York. Shipped to the western

part of the United States, cryolite would cost about \$35 per long ton, or 1.5c. per pound.

In the electrolysis of alumina in the electric furnace, the oxygen given off at the anode attacks the carbon electrode with the result that about 0.7 lb. of carbon anode is consumed per pound of aluminum produced. In France and Germany these electrodes are manufactured at the larger aluminum plants from coke at the cost of 2.5c. per pound. The cost of manufacturing them in the West with coke costing, at a minimum, \$10 per ton, or charcoal at \$8 per ton, would be about 5c. per pound. Electrodes could be brought from the East or Europe at a cost of 5c. to 6c. per pound f.o.b. San Francisco.

The cost of other materials, such as caustic soda and fluorspar, will not be discussed here, as the amount used is small, and their cost has been included in other estimates.

Labor costs in the West for manufacture of aluminum will be higher than in the Eastern States or Europe. The average wage for laborers at aluminum plants is \$1 per day in Europe, and \$1.50 in the Eastern United States and \$2.50 in the West, outside of mining camps.

Power.—The manufacture of aluminum requires direct current, so that the use of high-tension alternating currents, as delivered by Western power companies, would mean the transformation of this high current down to a lower voltage for use in a motor-generator set. Allowing for transformer, motor, generator, and line losses, this would result in the delivery to the furnace from the direct-current generator of about 85 per cent. of the energy delivered to the transformers. If we assume that the plant would be operated on a flat rate of only \$10 per horsepower-year for 11,000-volt alternating current, the cost of power at the electrolytic furnace, on the basis of 100 per cent. load factor, would be \$11.76. While it is sometimes asserted that an industry like the manufacture of aluminum can be operated on a 100 per cent. load factor, this cannot be used as a safe figure in calculating costs. The manufacture of aluminum, however, will maintain a higher load factor than some other similar industries, as for instance electric-furnace manufacture of iron and steel, because of the small units employed, and the ease of replacing a broken-down unit with a new one without much additional capital outlay caused by keeping reserve units. The largest aluminum furnaces do not require over 150 hp. for operation, while electric iron and steel furnaces have a capacity of from 300 to 12,000 hp. Another point, influencing the load factor maintained, is the market for the product manufactured. This is especially true for a new concern which has to build up its market. When buying power on a flat rate instead of by the kilowatt-hour, with a variation of load factor say between 85 and 100 per cent., any curtailment of production due to market conditions will result in a heavy charge to the electro-metallurgical company for power which it is not using. For this

reason, a load factor of 90 per cent. is assumed in calculating the cost of power. This would increase the cost of direct current at the aluminum furnace to \$13.06 per horsepower-year, or 0.2c. per kilowatt hour.

Cost of Production.—The following estimate of cost of production of aluminum is based upon a plant requiring 25,000 hp. or more, in units of 100 to 150 hp.

Cost of Production of Aluminum per Ton (2,000 lb.) with Conditions as above Stated

2 tons of alumina at \$28.75 per ton.....	\$57.50
200 lb. of cryolite at 1.5c. per pound.....	3.00
1,400 lb. electrodes at 5c. per pound.....	70.00
Other fluxes, etc.....	10.00
28,000 kw-hr. at 0.2c. per kilowatt hour.....	56.00
Labor.....	70.00
Repairs.....	10.00
Amortization, depreciation, 5 per cent. each.....	18.00
Interest, 6 per cent.....	10.00
General.....	20.00
Total.....	\$324.50

per ton, or 16.22c. per pound.

Allowing \$15 per ton for freight to New York and \$20 per ton for marketing expense, the total cost per ton f.o.b. New York would be \$359.50 per ton, or 17.98c. per pound, or in round numbers, 18c. per pound.

Market.—The average prices for No. 1 ingot metal at New York during 1913, generally for spot metal rather than future deliveries, are stated below:

Months	Cents per Pound	Months	Cents per Pound
January.....	26 $\frac{1}{8}$ to 26 $\frac{3}{8}$	July.....	23 $\frac{1}{8}$ to 23 $\frac{1}{2}$
February.....	25 $\frac{5}{8}$ to 26 $\frac{1}{8}$	August.....	22 $\frac{7}{10}$ to 23 $\frac{1}{10}$
March.....	26 $\frac{5}{10}$ to 27 $\frac{3}{10}$	September.....	22 to 22 $\frac{3}{8}$
April.....	27 to 27 $\frac{3}{8}$	October.....	20 to 20 $\frac{1}{10}$
May.....	26 $\frac{3}{10}$ to 26 $\frac{3}{4}$	November (first half)....	19 $\frac{1}{2}$
June.....	24 $\frac{3}{4}$ to 25 $\frac{1}{4}$	November (last half)....	19 $\frac{1}{8}$ to 19 $\frac{3}{8}$
		December.....	18 $\frac{7}{10}$ to 19 $\frac{3}{10}$

The tariff act enacted in October, 1913, placing a duty of 2c. per pound on ingot aluminum instead of the previous duty of 7c. per pound, resulted in the price falling in October of that year to from 20c. to 20.5c. The above quotations are indicative of transactions in the open market, especially as made by dealers in foreign aluminum. The average price of contract aluminum was 1c. to 2c. per pound less than the prices given. Hence it is evident that in order to avoid a shutdown in dull times, a company manufacturing aluminum in the western part of the United States should be able, if necessary, to sell its aluminum in the New York market at a price of 18c. per pound.

Aluminum manufactured in the West would at present have to com-

pete with a domestic production of 45,000,000 lb. of aluminum, and imports of about 30,000,000 lb. It is asserted that the domestic production of aluminum will in the future be increased by at least 20,000,000 lb. per annum, due to the Southern Aluminium Co. starting its plant, so that inside of two years all of the present imports except 10,000,000 lb. will be supplied by domestic producers. It is asserted also that the Aluminum Co. of America contemplates doubling its present capacity at a plant in Tennessee. If this be done, it will eventually add a possible production of 40,000,000 lb. per annum, or a total future output of 30,000,000 lb. more than the present market demands. Thus it is clear that any company starting business now will have strong competition to meet several years hence, or, in other words (since its plant could not be completed for several years), from the very start of its operation. This competition would exist in spite of the fact that the consumption of aluminum in the United States has increased at the average rate of about 7,000,000 lb. per annum for the last five years. While at this rate of increase of consumption the contemplated increased output would no more than supply the domestic demand, it is not probable that the consumption will maintain quite as high a rate of increase. Moreover, by reason of the decreased tariff, foreign manufacturers will also be much more active in the New York market hereafter.

Conclusions.—In view of the above statements, we are of the opinion that the manufacture of aluminum in the western part of the United States would not prove profitable.

Even if electric power could be had at \$10 per horsepower-year, the high freight charges on raw materials and product and increased labor costs over Eastern wages are such as to raise the cost of production so high that in periods of depressed market it would about equal the lowest selling price. In this connection it should be remembered that by reason of decreased output manufacturing costs invariably increase in times of business depression. Since such a plant would buy power on a flat-rate basis, any decrease in output would result in a greater increase of unit cost, due to the continuing charge for power regardless of the amount used.

A study of the preceding data shows that freight rates on raw materials, supplies, and product would contribute largely to the cost of the production of aluminum in the West. While India bauxite might be shipped to the West very cheaply, little is known about these deposits, and their development is in the distant future. An example of the cost of materials due to freight is in electrodes. They can be purchased in the East at 3c. per pound. On the basis of 0.7 lb. consumption of electrode per pound of aluminum, the cost of electrodes at 5c. per pound increases the cost of aluminum by 1.4c. per pound.

Besides the higher labor costs due to increased wages in the West, technical men, such as chemists and engineers, also receive a higher rate of pay.

At the low price of aluminum in 1913, a company manufacturing aluminum in the West could about "break even" if it sold its product. At a price of 20c. per pound a profit of 2c. per pound could be made. A production of 10,000,000 lb. per annum would require a plant costing about \$4,000,000. This production would give a profit of \$200,000 per annum, or 5 per cent. on the actual investment. This is generally considered much too low a margin of profit for safe investment, since unforeseen conditions might arise to increase the cost. Estimates previous to operation are usually found in practice to be too low. It is probable that the Eastern producer could sell at a profit at 18c. per pound; since it has been estimated that the cost of production of aluminum at Niagara Falls is about 15c. per pound.

A company now producing aluminum, with a market for its product and an intimate knowledge of the industry, could possibly establish a manufacture in the West; but we see no reason why such a company should go to the West for such a purpose at the present time, or in the very near future, at least so long as it is able to secure power near the ore and the market, as at the present time. Apparently, an established company owning its bauxite supply and having a market, is the only sort of a concern which would be able to operate a plant in the West without losing money, and it could not make as much there as in an Eastern plant.

Pig Iron.

Raw Materials and Labor.—The raw materials necessary for the manufacture of pig iron in the electric furnace are iron ore (either hematite or magnetite), limestone, and charcoal or coke. There is no choice as to the kind of iron ore, as the electric furnace will handle either hematite or magnetite. In regard to the limestone, it is sometimes considered advisable to calcine the limestone before use in the electric furnace, in order to save the energy necessary to remove the carbon dioxide from the limestone in calcination; but the use of calcined limestone is not advisable because of the fine material added to the charge in this way. Charcoal is preferable to coke in the electric-furnace manufacture of pig iron, because the energy consumption is smaller (because of the possibility of using the shaft type of furnace with charcoal), and operation more steady. Coke can be used, however, in the rectangular type of furnace without a high shaft.

Iron ore of good grade is more or less plentiful throughout the Western States, but unfortunately the cost of transporting it to a point where it could be utilized might prohibit its use. Moreover, it is estimated that Chinese iron ore containing 60 per cent. of iron can be now laid down at Pacific Coast ports at a cost of about \$5 per ton. Such being the case, we will assume that the cost of iron ore would be \$5.

Limestone deposits are also more or less abundant throughout the

West, and so the cost of lime as a flux in the production of pig iron would probably not be prohibitive.

As already observed, charcoal or coke could be used as a reducing material. At the starting of a plant, charcoal would be the preferable material, but if the plant assumed great proportions, say requiring much over 100 tons of charcoal per day, we believe that it would be necessary to use coke, on account of possible scarcity of charcoal. For the purpose of this paper, we will assume the cost of the reducing agent as not less than \$10 per long ton, or \$3.33 for about one-third of a ton, required for reduction per ton of pig iron produced.

Labor requirements are the same as in the blast-furnace manufacture of pig iron, and a minimum wage of \$2.50 per day of 8 hr. is assumed.

Power.—Power will be considered as costing at the furnaces \$11.70 per horsepower-year, or 0.18c. per kilowatt-hour, allowing for transformer losses in reducing the voltage to from 40 to 110 volts.

Cost of Production.—The following estimate is based upon an annual production of 50,000 tons of pig iron. The plant would consist of five electric furnaces of 3,000 hp. each, the whole plant requiring 18,000 hp., including 500 hp. for various uses outside of the furnaces.

*Estimated Cost per Long Ton of Producing Pig Iron in the Electric Furnace,
with Conditions as above Stated*

1.6 tons of iron ore at \$5 per long ton	\$8.00
0.33 long ton of charcoal or coke at \$10	3.33
0.25 long ton of limestone at \$1.75	0.44
10 lb. of carbon electrodes at 5c.....	0.50
2,400 kw-hr. at 0.18c.....	4.32
Labor.....	5.00
Maintenance and repairs.....	0.50
Amortization, depreciation, at 5 per cent. each	1.70
Interest at 6 per cent.....	1.02
General.....	1.40
Total.....	\$26.21

Based upon the above estimate, the cost of producing pig iron by use of the electric furnace would be about \$26 per long ton. As the market for this pig iron would be entirely upon the Pacific Coast, it would have to compete with iron brought from the East and with foreign iron.

Market.—The market for this pig iron manufactured in any of the intermountain or Pacific Coast States would largely be a local one. At the present time, this market would probably not be very great, since with pig iron selling on the coast at from \$20 to \$25 per ton, there is little incentive to use it in foundry work, when scrap iron can be purchased at a much lower price. There is no large steel plant on the coast which would be a consumer of pig iron.

Another important factor lies in the cheapness with which pig iron from England, China, and India can be laid down upon the Pacific Coast. Pig iron from any of these countries could be delivered at Pacific Coast ports for from \$18 to \$20 per long ton. With the Panama Canal open, it is now possible to lay Eastern pig iron down on the Pacific Coast for about \$18 per long ton, so that the foreign or Eastern producer could considerably undersell the Western manufacturer.

The high cost of production is largely caused by the high cost of iron ore, which is about double that paid by Eastern blast furnaces, which are producing pig iron for about \$10 per long ton; and also by the high labor cost, which is about 70 per cent. more expensive than Eastern labor. The cost of reducing material is also high as compared with the cost of Eastern coke at the furnaces, in spite of the fact that the electric furnace uses only one-third of the amount of coke used by the blast furnace. It would therefore seem that the cost of production alone would prohibit the success of an electric-furnace pig-iron industry in most of the Western States.

To assure commercial success, the plant should be able to sell its product in the coast market, if necessary, at as low a figure as \$18 per ton. Eastern or foreign pig iron can be laid down at that cost. It is also quite likely that the market for pig iron, with the price as high as \$20 per ton, will not grow very rapidly, as it is cheaper to use scrap iron to make castings.

Steel

There are two forms of steel manufacture in which the electric furnace has been used: (1) cold scrap iron and steel of either inferior or high-grade quality are melted and refined in an electric furnace with the production of steel of the highest grade and equal to the best crucible steel; and (2) molten steel, the product of either the acid or basic converter, or of the acid or basic open-hearth furnace, is super-refined, or made into alloy steel, in an electric furnace. The steels thus made may be cast into ingots, or directly into various shapes. It has been proposed to use the electric furnace for the manufacture of steel from molten pig iron; but with pig iron at \$20 per ton, the cost would be prohibitive. Steel made from molten electric-furnace pig iron would cost at least \$34 and probably over \$40, while that made out of Eastern pig iron, which would have to be melted, would cost at least \$30 and probably \$35, and so could not compete with Eastern steel, which can be sold in the West for about \$30.

As the high cost of pig iron prohibits the establishment of a tonnage steel plant, we will consider only an electric-furnace plant for the production of high-grade steel castings and shapes, and bar steel.

Raw Materials and Labor.—The principal raw material used in the

electric-furnace manufacture of steel is scrap steel. While some scrap cast iron could be used, most of the material melted should be steel or wrought iron. Iron turnings, which in the open hearth are not especially desirable on account of oxidation losses, are about the most adaptable material for use in the electric furnace. There is not the high oxidation loss in the electric furnace that there is in the open hearth. Any scrap material used in the electric furnace must be small in size because of difficulty in operation of a furnace on large scrap iron, due to short circuits. A large part of the turnings produced in the various foundries and shops throughout the West go to waste at present, and could be obtained cheaply.

Power.—In the electric-furnace manufacture of steel, electric energy is used at about the same voltage as in the production of pig iron or ferro-alloys; but the electric steel furnace cannot maintain as high a load factor as the electric iron-smelting furnace or the ferro-alloy furnace. For example, if it be necessary to transform an 11,000-volt current down to 40 or 100 volts, a transformer and line loss of 5 per cent. will occur. This, on the basis of 100 per cent. load factor and with power at \$10 per horsepower-year, would make the power cost \$10.52 at the furnace. Owing to the intermittent nature of the electric-furnace process in steel manufacture, a load factor of over 80 per cent. could be maintained only with difficulty. The power cost would then be \$13.15, or 0.20c. per kilowatt-hour. If a lower rate were made on such power, an electric steel furnace could use peak power to advantage, on account of the intermittent nature of the process.

Cost of Production.—The following estimated cost of production of electric-furnace steel is based upon an annual production of 25,000 tons and the utilization of 4,000 hp. There are so many combinations of furnaces of different sizes with which a plant could be equipped that we will not attempt to specify them. The estimate is based upon steel cast into ingot form.

Cost of Production of Steel in the Electric Furnace in the Western United States

1.1 tons of scrap at \$15 per ton.....	\$16.50
Slag materials.....	1.00
Ferro-alloys.....	1.00
800 kw-hr. at 0.20c.....	1.60
Labor.....	2.50
Maintenance and repairs.....	2.40
20 lb. of electrodes at 5c.....	1.00
Amortization and depreciation at 5 per cent. each.....	1.50
Interest at 6 per cent.....	0.90
General.....	1.00
Royalty.....	0.50
Total.....	29.90

With conditions as above stated, the cost of production of ingot steel in the electric furnace would be about \$30 per long ton, to which would have to be added the freight rates to the point of market or delivery.

Market.—There is very little market for billet and ingot steel on the Pacific Coast, and at the present time a tonnage steel plant manufacturing ingots for rails and heavy steel would probably not be able to compete in the market with Eastern or foreign products. But there is a considerable demand for small shapes; and a plant in the West casting steel into such shapes might possibly be able to dispose of its product; but a large part of the steel shapes used in the West belong to machinery manufactured in the East, and most of this steel comes from the Eastern States.

Conclusions.—Considering the conditions above stated, it does not look as if the manufacture of tonnage electric-furnace steel in the West would be profitable, as steel in the form of ingots, billets, rails, or structural shapes can be laid down so cheaply on the Pacific Coast from the Eastern States or foreign countries as to render impossible the profitable manufacture of this steel on the Pacific Coast at the present time. While a small electric-furnace foundry, manufacturing special steel shapes and using 500 to 1,000 hp., might be successful, it would not amount to much as a large-scale steel producer. Tonnage steel cannot be made at a profit on the Pacific Coast at the present time, also due to the high cost of producing pig iron, and for this reason it would be impossible to successfully operate a self-contained iron and steel plant, using either electric or combustion furnaces.

Copper

As already observed, the electric smelting of copper ores is entirely in the experimental stage. Before it can be practiced commercially, much money and time must be spent on experimental work. While we believe it to be feasible under favorable conditions, the process has not yet reached a stage of development that would warrant us in regarding it as fully worked out. When some of the Alaskan deposits have been more developed, electric copper smelting may prove profitable at points along the Alaskan coast, since the ore in this case could be hauled largely by water. As the process itself is not fully developed, we do not consider it worth while to give an estimate of the cost of smelting copper ores by electricity.

Zinc

While the electric smelting of zinc ores is more advanced than that of copper ores, it has not been yet proved a commercial, or, in fact, a technical success. Such being the case it cannot, at present, be seriously considered as a consumer of power. However, the process of electric

zinc smelting will doubtless be more fully developed in the near future, and it is quite likely that the electric smelting of zinc ores will consume considerable power, but at present, as in the case of copper, we do not consider the process sufficiently developed to warrant estimates of cost.

IV. GENERAL CONCLUSIONS

1. The market for hydro-electric power for electro-metallurgical industries in general is not great; and under the best of conditions the part consumed in electro-metallurgical plants would be a very small proportion of the total hydro-electric power that it is possible to develop in the Western States.

2. Although the outlook at the present time is not favorable to the establishment of extensive electro-metallurgical industries, we are nevertheless of the opinion that such industries will, in time, be established, but perhaps along other lines than at present, that is, other than the production of aluminum, ferro-alloys, pig iron, steel, etc. It is also doubtless true that the electric furnace will in time be quite generally used for the local production of steel castings, but an extensive use of the electric furnace for this purpose will not involve any great consumption of electric power.

Therefore, if our analysis of the situation be correct, is it reasonable to expect that the hydro-electric power companies may ultimately be able to dispose of a great portion of their surplus power by reason of the development of new electro-metallurgical industries? We believe they will, and for this reason:

At the present time the metallurgy of the non-ferrous metals is rapidly changing. The processes which were suited to the treatment of non-ferrous ores five or ten years ago are at the present time not satisfactory, because the non-ferrous metallurgical plants of the country are called upon to treat ores of a lower grade, and also more complex than formerly. Such being the case, processes must be devised which will meet these requirements. However, we do not for a moment imagine that electro-thermic or electrolytic processes will prove to be the only solution of the many problems which are at the present time confronting the non-ferrous metallurgist. Although such processes may greatly assist in solving these problems, we are of the opinion that further research work will indicate other solutions. This is true, especially as regards hydro-metallurgical processes, because the profitable treatment of the low-grade and complex ores above mentioned requires the use of cheap reagents. Such being the case, the hydro-metallurgical treatment of such ores may bring about the establishment of an electro-chemical industry for the production of the necessary reagents and thus indirectly the metallurgical industry may bring about an extended use of hydro-electric power.

Whether this will prove to be the case or not, remains to be seen. It can only be determined by a careful investigation of the subject, and by extensive research, having for its object the finding of new uses for electricity in metallurgical work.

DISCUSSION

LAWRENCE ADDICKS, Douglas, Ariz.—I think papers of this character, while general in nature, are of particular value and interest, because when we figure on competing with the East in electrometallurgical industries, on account of the cheap power on the Pacific Coast, we have to take two or three things into account. In the first place, power very rarely amounts to more than 20 per cent. of the total cost of an operation, even though it be electrometallurgical, and I think it is the common opinion among those who are not familiar with those particular industries, that it amounts to more than 50 or 60 per cent. Take copper refining; we may say that it is 15 per cent. of the total cost. In the second place, around New York City, with 100 per cent. load factor and a reasonably large load, say 10,000 kw., we can generate from a certain class of coal 1 kw.-hr. for about 0.3c. I might say, however, that 0.3c. does not include overhead charges, interest on the investment, etc. The third factor we have to consider out here is that any product to be salable must command a retail market. It would be useless to produce wire bars here on the Pacific Coast because there is no place to roll them. If we were going to establish a copper refinery, that part of its output which goes into wire bars would have to be taken care of by building a rolling mill alongside of it to turn it into wire and finally make a product that is salable. I do not think, however, that Mr. Lyon need have excluded copper refining from his industries. There are perhaps 50,000,000 lb. of copper a month refined in this country on a custom basis. I think the smelting companies would be very glad to build refineries here if conditions warranted it. The recent extensions of the Great Falls refinery and of the Tacoma refinery in the last few months give evidence of that.

CARL H. BOOTH, Chicago, Ill.—I agree in a general way with the features of Mr. Lyon's paper. I have read it with very great interest. It seems to me the real reason for the lack of development of electrometallurgical processes on the Pacific Coast is largely the lack of a market. I am not claiming to have a great knowledge of conditions out here, excepting those I have personally observed in the last few weeks, but my general conclusions are that in the one field of electric steel melting and refining, regardless of the low cost of melting, the lack of a proper market will always be a severe handicap to any operation of that kind on the Pacific Coast. Even with cheap power and low-cost materials, the

limited tonnage available is a distinct disadvantage. If the steel made here must go East to find a market, the cost of freight rates, and the fact that power can be produced almost as cheaply in the East, will more than offset any other advantage; so that my general thought is, being especially interested in the line of steel metallurgy, that the chief drawback to the manufacture of a heavy tonnage of steel on the coast is the lack of a market.

Apparently the authors of this paper are not entirely familiar with some of the recent developments in the electric steel melting field. Electric furnaces are today being built which produce steel cheaper than the fuel melting furnaces for the same tonnage. These facts have been well worked out and carefully checked for a period of two years. Such furnaces are today producing steel at \$18 a ton with current at $\frac{3}{4}$ c. per kilowatt-hour, and materials at \$11 per ton. The upkeep cost is low and the efficiency of the furnace is so high that it has been practical to obtain low-cost operations, and to use electric power and still compete with fuel melting. With this one point of difference, the cost of melting with the electric steel furnace, I thoroughly agree with Mr. Lyon's conclusions. I would call your attention especially to p. 843, where he gives a schedule of the operating cost of an electric steel furnace. With current at the price he gives, steel could be produced for about \$18 a ton, so that there is a difference of at least \$12 a ton in Mr. Lyon's figures.

It makes no difference how cheap you can make the steel out here; if you have no market for it, it is hardly possible to do much with the product. Even if steel is substituted for iron, with the increased tonnage that might be obtained still I can see no hope for developing a large industry here in the immediate future.

J. W. BECKMAN, San Francisco, Cal.—I take pleasure in disagreeing with the attitude that both Mr. Lyon and Mr. Booth assume in regard to possibilities on the Pacific Coast. I think it is undoubtedly true, if you only look to the near future, that in the main they are correct in saying that there are small markets only, but this is a new country, and we are practically in the same position here as the East was before the Civil War. Before the Civil War the East would not do any manufacturing because they thought they could not compete with England. As they could buy what they needed from England they saw no reason for manufacturing. The East has changed! It has been shipping for many years, and for a long time to come will ship manufactured products to England. Here on the Pacific Coast everybody apparently is looking to the East for their markets. I have often heard said: "What is the use of manufacturing out here, they can beat us in the East?" I think that is a wrong attitude for the people of the Pacific Coast to take. We are favored with enormous quantities of water

power here practically at tidewater. There are a number of opportunities right on tidewater, and a number just a few miles away from tidewater. We have, therefore, the same power facilities on the Pacific Coast to produce electrochemical materials that Sweden and Norway have, especially Norway, and the same opportunity for shipping these materials to the world's markets. My feeling is very strong that the future of the Pacific Coast does not lie east from here, but west and south. We have over half of the world's population facing the Pacific Coast. We should take care of the electrochemical needs of that population and we will do so, provided we look at matters in the right way. If we look only to the east we are certainly going to let everybody in the East, as well as Europe, take care of those markets that legitimately belong to us. Therefore, I feel that the broad-minded view is not to look forward to what this generation can do, but build for generations ahead and start doing so now. That is a duty resting on the present generation on the Pacific Coast. There is an enormous future for this coast. I think a great part of the people on this western coast do not appreciate the opportunity this section of the country affords. We have the Chinese and Japanese markets of unlimited possibilities; we have the northern part of Siberia, that we do not know so much about; there are also markets to be had in South America and Australia, and we might reach for markets even as far as India, and in years to come I believe my attitude will be proven correct. I think this attitude of belief in the possibilities of this coast and its future is psychologically the right one to hold. Mr. Lyon does not seem to have taken into consideration the fact that we can get aluminum oxide at a very low figure here, from a deposit in Utah that is being worked now, and great quantities of practically pure aluminum oxide are obtained from it. It is claimed that the product is obtained so pure that in many cases there is no need of further refining. A reasonable freight rate has already been established by the Western Pacific R. R. for this aluminum oxide from those deposits to the Pacific Coast. This aluminum oxide is obtained as a by-product at a very low price, and it seems as though there is a big possibility for making aluminum on the coast from this raw material.

Mr. Lyons says that all our electrodes are shipped here from the East. We have here on the Coast one very large and permanent source for the manufacture of electrodes in what is generally known as lampblack. In the making of volatile products from oil there is always produced quite a quantity of exceedingly fine carbon with some heavy oil mixed in. In many cases this has been used as fuel under boilers, because of no other use for it. From my knowledge of electrodes I believe that there is a possibility of using this practically pure carbon in the making of electrodes, and using them in aluminum manufacture and in electric furnace work.

Pig-iron manufacture is a very old process and no radical change has taken place in its manufacture until comparatively recently. Is it not possible that such an extremely old process will be substituted in time by an absolutely new process utilizing new agents in it? Here on the Pacific Coast, for example, we have an abundant supply of natural gas and an abundance of fuel oil; we also have an abundance of cheap electric power. Couple with these an abundance of cheap iron and it seems as if there must blossom forth a large iron industry.

Along the coast there are great deposits of iron ore. About 200,000,-000 tons are now located, running 64 per cent. of iron. It is of interest to note that the United States Steel Corporation is running at the present time on 56 per cent. iron ore.

Why should we not look forward to the utilizing of new agents and methods in place of the old processes for the manufacture of pig iron?

Another point that has impressed me in connection with Mr. Lyon's paper, and to which I have not referred, is the question of treating refractory ores by electrolysis in fused electrolytes. I believe it might be possible by such a process to partly separate the different metals present in the ores.

LAWRENCE ADDICKS.—Mr. Beckman has put the problem on its true basis. If we did not have cheap power here, the whole matter would be impossible. Of course, in regard to aluminum, we all of us have an aluminum mine in our back yards, but the difficulty has always been to get a deposit of value. Will Mr. Seaver tell us what the refracting companies have done in the direction of using alumina?

KENNETH SEAVER, Pittsburgh, Pa.—I feel that I am hardly qualified to speak with any authority about what has been done by the refractory companies in this field. You are all familiar with a brick made from hydrous oxide of aluminum. I speak now, in the first place, of a content of about 56 per cent. in the brick, and another of about 77 per cent. of alumina. The characteristics of the two bricks vary rather widely. Those two are probably the extremes of alumina commercially valuable to the refractory industry. The 56 per cent. aluminum content is used for fire-clay brick, but it will withstand higher heat. We are here considering alumina as neutral, although we are familiar with its action under varying chemical conditions, and this 56 per cent. content aluminum brick has been used where reasonably high heat was encountered. The other material running as high as 77 per cent. in alumina content has thus far been used as a substitute for magnesia brick, or more often termed magnesite brick, in the open-hearth and similar processes. It has been giving excellent results, but those results in the light of years of experience, should not be taken for more than they are worth. It is not yet safe, in my opinion, to say that it is a substitute. I believe that

it is. Under normal conditions it probably cannot be produced at the price at which magnesia brick was produced. This particular refractory has been used in the port end of a furnace, forming a bulkhead of that one particular open-hearth furnace, for 365 heats, at which time the furnace was necessarily taken off to install a new roof. At that time the opposite port, which was installed with magnesia brick alone, was in a worse condition. The indications are that this 76 per cent. alumina brick will give equally good results. On the other hand, I feel, bearing in mind the experience of years, that a thing of this kind must be tested out under every conceivable condition before one can make a flat statement on the subject. I believe it shows great possibilities, however.

J. W. RICHARDS, South Bethlehem, Pa.—I wish to speak on several matters connected with the subject of this paper. The first is the possibility of producing aluminum on this coast. Mr. Lyon considers the cost of aluminum here and shipping it to New York. I believe that if such a plant were established here the material would not be shipped away, but would be used to supply the local market, and also the Japanese market which is a considerable one at the present time. The principal raw material is bauxite, or its equivalent, and the question is whether it can be found west of the Rocky Mountains. If the byproduct alumina from the Maryvale alunite deposits in Utah could be brought here at \$5 per ton, that might put a different aspect on the matter, or, if bauxite were to be found on the Pacific Coast there would be a possibility of establishing works here to supply the local market and the Japanese or other foreign markets bordering on the Pacific Ocean.

In regard to the consumption of steel on this coast, in the paper by C. C. Jones of Los Angeles, it is stated that the consumption of steel on the Pacific Coast is nearly 3,000,000 tons per year. That is not an insignificant market, although I got the impression from Mr. Lyon that he considered the local market here was not worth considering. That is a respectable proportion of the output of the United States. If there is such a market here, and the statements made as to the cost of making steel in a properly designed electric furnace are to be relied upon, there is a distinct possibility of the Pacific Coast furnishing its own supply of steel castings and possibly some manufactured shapes. We must give, too, a proper consideration to the practical importance of the ferro-alloy industry which has already begun operations on the Pacific Coast. Furnaces in Shasta county which were producing pig iron are operating on ferromanganese, ferrosilicon, and ferrochromium, and they are at present operating successfully. There is a possibility, it seems to me, of firmly establishing those industries on this coast.

J. W. BECKMAN.—I take exception to what Professor Richards has said about ferromanganese. It is not possible to establish the ferro-

manganese industry here with an electric furnace. It might be with a blast furnace but not with an electric furnace.

D. A. LYON, Salt Lake City, Utah.—I would like to call attention to the fact that in this paper we do not state that it will never be feasible to establish electrochemical and electrometallurgical industries on this coast, and that the establishment of these industries should not be considered, but have stated our views as regards the possibility of electrometallurgical industries becoming large consumers of electric power here on the Pacific Coast at the present time.

As to the tonnage of steel used on this coast; the day before yesterday I was talking to a prominent iron and steel metallurgist of this country in regard to the possibility of producing steel here in the West, and he remarked that it would be practically impossible for any one plant to turn out all the different shapes that would be required to meet the demand, and that while the total tonnage of steel used on the Pacific Coast is comparatively large, there is not a large tonnage used of any particular shape or size. Such being the case, there is no demand at the present time for a plant producing a large tonnage of iron and steel. I think the gentleman was right and must therefore disagree with those who believe that there would be a market for the output from a large plant at the present time.

Method of Making Mineralogical Analysis of Sand*

BY C. W. TOMLINSON, M. A.,† MINNEAPOLIS, MINN.

(San Francisco Meeting, September, 1915)

INTRODUCTORY

THE analyses which have been made by the writer according to the method described below were made as part of Professor Withey's investigation of the concrete aggregates¹ of Wisconsin, in the hope of throwing light on the extent and nature of the influence of the mineralogical constitution of aggregates upon the properties of the concrete and mortar made from them.

This method was found most satisfactory after four weeks of experiment and modification. It attains an accuracy quite sufficient for the purpose of the analysis, and is fairly rapid. Mineralogical analyses may be thus made at a cost considerably less than that of chemical analyses of a similar degree of completeness.

The results obtained by this method should not vary in any item by more than 1 per cent. of the total weight of the sample, and may be made very much better than this by the exercise of greater care in the microscopic counting (see below). The accuracy of the hand sorting is of course dependent upon the experience and carefulness of the analyst.

After a month's practice, the writer was enabled to turn out these analyses at the rate of four in 33 hr. of working time. It was found most economical to handle four samples at once.

The samples analyzed weighed from 290 to 340 g. each. The same method could be applied to larger or to smaller samples, within reasonable limits.

The following list covers the necessary apparatus and equipment:

- 1 good balance capable of weighing accurately to tenths (or, better, hundredths) of a gram, and of handling quantities up to 200 g. in weight.
- 1 Westphal balance, registering densities to two or three decimal places.
- 1 Meker or Bunsen burner, and gas supply.
- 1 nest of sieves, 10, 20, 40, and 100 meshes to the inch.

* Developed at the University of Wisconsin in June and July, 1914, by C. W. Tomlinson, under the direction of M. O. Withey and A. N. Winchell.

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¹ The *aggregate* comprises all of the material entering into the composition of a block of concrete, other than cement and reinforcement. Gravel, sand, crushed stone, and slag often constitute concrete aggregates.

- 1 binocular or petrographic microscope (not necessary for rough work).
 1 good-sized hand lens of moderate power.
 1 rubber laboratory apron.
 1 pair rubber gloves.
 Chemicals: potassium iodide and mercuric iodide (see following pages).
 Bottles to hold heavy solution.
 1 1,000-cc. wash-bottle.
 6 Harada tubes, if these are used.
 Beakers, as follows: two 500-cc.; three 200-cc.; five 100-cc.; three 50-cc.
 Casseroles: two 4-in.; two 2-in.; one 6-in.
 1 12-in. evaporating dish.
 4 4-in. glass funnels, for filtering.
 3 8-in. glass rods, $\frac{3}{16}$ in. diameter, for stirring.
 8-in. and 6-in. filter paper.
 5 yd. toweling.
 1 tripod support for evaporating dish.
 2 ring stands with 2 rings each, and clamps for supporting separation beakers.
 1 5-in. square of wire screening.
 1 counting sheet (see paragraph on "Microscopic Analysis").
 2 by 3 in. and 3 by 4 in. envelopes for filing separated sands.
 (Envelopes used for steel pens are suitable).
 Larger envelopes for filing separates of each sample.

OUTLINE OF METHOD

The various steps in the process of a complete analysis of an average or complex sand, with the time (figured for four analyses conducted at once) required by each, are as follows:

	Hours	Minutes
Sizing	1	15
Weighing of sizes	45
Preparation of heavy solution	2	..
Heavy-solution separations	5	..
Hand sorting	12	20
Weighing of separates	3	..
Microscopic analysis of heavy solution separates ..	3	30
Computation of analysis and preparation of report.	4	30
<hr/>		
Total for four analyses	32	20
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Time charged to each analysis	8	5

The chief saving in time from carrying on several analyses at once is in the heavy-solution work. The heavy-solution separations and the hand sorting may be carried on together, the latter filling the intervals during which the solution is being evaporated.

Each of the several steps is described in detail below.

Sizing

A separation of the sample into several sizes is advisable for three reasons: (1) The coarser material can best be classified by hand sorting;

(2) great range in size of grain reduces the accuracy of the heavy-solution separations; and (3) the resulting information is of value, since the composition of the sand differs in the several sizes according to definite principles. Material too coarse to pass a 20-mesh screen is apt to contain many fragments of phaneritic igneous rock not yet disintegrated into constituent minerals, and therefore of variable specific gravity. Moreover, material of this size can be rapidly sorted by hand, and can be more satisfactorily and completely classified than by specific gravity alone (heavy-solution separation); and the separates can then be *weighed*.

The sizing is conveniently done with a nest of sieves of 10, 20, 40, and 100 mesh. For accurate work each sieve should later be handled separately to perfect the sizing.

Weighing of Sizes

This is done for two reasons: To serve as a check upon the possible losses in manipulation and upon possible errors in the final weighing of separates, and to determine the care with which each size shall be sorted. Following the weighing, the percentage of each size in each sample is computed. If any group amounts to less than 1 per cent. of the whole, it may be neglected in the analysis. Each size is laid out upon a sheet of paper or (better) inserted in an envelope of appropriate size, and labeled with the number of the sample. If a standard set of sieves is used, it is not necessary to write down the size, as this is recognized at a glance—unless envelopes are used. Envelopes should be free from leakage, and should be sealed by folding the open end or side, not by pasting down the flap.

Preparation of Heavy Solution

An excellent classification of the sand which passes the 20-mesh sieve may be made by means of heavy solutions. A solution of density 2.9 will separate practically all of the dark silicate minerals from the others. One of 2.7 will separate quartz (2.65) from calcite (2.72) and dolomite (2.8 to 2.9). One of 2.585 or 2.59 will separate orthoclase from quartz and plagioclase. The first two of these separations are highly satisfactory, but the third (density = 2.585) is less so because of the wide range in density of chert and quartzite due to varying porosity. Most chert will float with the orthoclase. Most common igneous rocks will fall with the carbonates in the group between 2.7 and 2.9; but these are seldom abundant in the sizes of sand submitted to heavy-solution separation. Graywackes behave like the igneous rocks. Mica is not to be trusted, because its great surface area interferes with proper separation; the heavy solution has a high surface tension.

Probably the most satisfactory solution is that of Sonstadt and Thou-

let, which consists of potassium mercuric iodide in water. Its preparation, characteristics, and manner of use are well described in Johannsen's *Manual of Petrographic Methods*, pp. 519 to 521. It is not necessary to evaporate on a water bath, as there prescribed, if the analyst will keep fairly close watch of the solution as it nears its maximum density. Evaporation in a dish resting directly on a wire screen on a ring stand over a burner is much more rapid and produces exactly the same result. Allow the solution to cool before filtering.

The quantity of solution needed for handling four analyses at once—to conduct eight or ten separations simultaneously—is about 2,000 g. This requires about 1,080 g. of mercuric iodide and 920 g. of potassium iodide, total cost about \$12.

Too much emphasis cannot be laid upon the necessity for caution in handling the Sonstadt solution. A few drops on the fingers occasionally during one day may not produce any noticeable effect at the time, but is sufficient to cause a swelling of the flesh which may stop the work for one or two days. The fingers become distended and rigid, and the skin discolored and sore, sometimes developing a rash. A drop of Sonstadt solution under the finger nail will bring pain quite readily. Elsewhere, except on a cut or bruise, it produces no immediate effect, and is therefore so much the more dangerous. Rubber gloves should be worn during all operations involving this solution; and new gloves should be obtained as soon as any hole appears in the old ones.

A stain of Sonstadt solution on the clothing may be readily removed in water if applied at once, or in alcohol later; but a rubber apron is an excellent preventive of such stains.

When the required amount of solution has been prepared, evaporated to a high density (anything over 2.9 is sufficient), cooled, and filtered (filtering may be omitted at the judgment of the analyst), it should be brought to the density wanted for the first separation, by diluting with water. This is best done by diluting a part of it to a lower density than that desired, and then pouring the *heavier fluid into the lighter*; the lighter is placed in a beaker inclosing the pendant of a Westphal balance, and the heavier poured in from a casserole or other vessel with a good lip. As the proper density is approached, the introduction of the heavier fluid may be made drop by drop, and the desired density thus obtained with an accuracy of three or even four decimal places.

Heavy-Solution Separations

A full description of the nature and use of the various types of apparatus used for heavy-solution separations is to be found in Johannsen's *Manual of Petrographic Methods*, pp. 547 to 556. The simple Harada tube described on p. 549 is very convenient, and a single separation there-

with is quite satisfactory for the purposes of this work, without the repetition prescribed for more accurate results. An ordinary beaker may be used with practically as good results, however, according to Goldschmidt's method, described on pp. 548 to 549 (Johannsen); this has the advantages of cheapness of apparatus and wide range in size of beakers (to accommodate different quantities of sand); and, what is an even greater convenience, the beakers can readily be wiped dry.

Before pouring heavy solution, the density of which has been accurately adjusted to a desired figure, into a separation beaker, it is important that the interior of the beaker should be quite dry; for a few drops of water may lower the density of the fluid enough to impair the results of the separation seriously.

In making each separation, the beaker should be chosen of such a size as to accommodate the parcel of sand with about twice an equal volume of solution, to insure a clean separation in decanting the two groups of minerals. If the quantity of sand in one size is too great to be satisfactorily handled, it may be reduced by quartering; and the proportions of minerals found in the tested portion are then assumed to hold for the remainder also.

It is a waste of time to drain off the heavy solution from the sands by filtering, after the separation is completed. The two separate parcels of sand (one of less and one of greater density than the heavy solution) should be washed directly into two separate casseroles, and there thoroughly washed by repeated decantations. A 1,000-cc. wash bottle is of convenient size for this work: a smaller one requires refilling too often. All of the solution used in the separation goes into the wash water, and is subsequently evaporated to high density again, before the next separation is attempted. During the process of evaporation the analyst may busy himself with the hand sorting of the coarser sizes of the samples with which he is working; though he should keep fairly careful watch of the progress of the evaporation.

As already remarked, the writer has found it economical to handle four or more samples at once. If four are taken, and there are two sizes from each sample to be submitted to classification by heavy solution (it is often the case that the "finer than 100-mesh" group is too small in quantity to warrant analysis), there will be eight parcels to be so treated, four of size 20 to 40 (passed 20-mesh, but not 40-mesh screen) and four of size 40 to 100. If the "finer than 100-mesh" group is to be treated in each case also, there will be 12. The requisite number of beakers (common or Harada) should be selected, of sizes appropriate to the sizes of the parcels of sand; and an adequate quantity of heavy solution should be prepared of the desired density. A 500-cc. beaker will hold enough (about 1,400 g.) for six or eight moderately large separations (50 to 75 g. of sand in each). One important economy of time effected by large-

scale operation is in the matter of preparation and adjustment of the heavy solution.

It is usually best to perform the separation at density 2.7 first. Nearly all sands contain some material on each side of this mark, and it is almost always the most effective separation. If either of the separates, the heavy or the light, in any sample then is very small in quantity, no further separation need be made of it; otherwise, the eight (or twelve) parcels of sand *heavier than 2.7* will be treated to another separation at 2.9, and those *lighter than 2.7* to one at 2.585.

For the same reason that the separation beakers should be dry before the Sonstadt solution is inserted, the sands should be thoroughly dry before a second separation is attempted. For this purpose each parcel, as soon as it has been cleansed of heavy solution, is washed from its casserole on to a pad of filter paper (three or four sheets in thickness) labeled with the number of the sample. The size and density of each parcel can be recognized at a glance, after a little practice. These may be left in the sun, or laid out on the table in the vicinity of the burner used in evaporating the wash water. If the latter method is used, the sands must be allowed to cool when dry before being subjected to another separation; as any notable rise of temperature may change the density of the heavy solution.

As in the first preparation of the heavy solution, the concentrate from the wash water of the first separation should first be allowed to cool, then filtered, then brought to the exact density wanted for the second or third separation. A 12-in. evaporating dish, a ring stand, and a Meker burner make an excellent combination for the evaporation.

Three separations of two sizes from four samples result in four separates from each size in each sample, or a total of 32 separates. When all are dry, they should be placed in separate envelopes (free from leakage), each appropriately labeled, as for example:

Sd. 8 20-40 D 2.585-2.7

Hand Sorting

The materials in each sample which do not pass the 20-mesh screen are again separated for convenience by a 10-mesh screen. If the quantity of plus 10-mesh gravel is less than 50 g., or that of 10-to-20-mesh sand less than 10 g., the entire parcel may be sorted by hand. For larger quantities it is desirable to quarter down the parcel until a representative portion of reasonable size is obtained. It is somewhat risky, however, to sort much less than 10 per cent. of a parcel and to assume that the composition of the sorted portion then represents accurately that of the whole. The time required for hand sorting depends wholly upon the extent to which this assumption is carried, and the ease and rapidity

with which the analyst can recognize and classify the grains. One little trick which the writer has found to save time is this: Do not pick up each grain with forceps and drop it in the appropriate pile, but separate one kind of mineral or rock from the entire parcel (or part of it which is to be sorted) first, then another, etc.; simply using a lateral motion *on the paper*, sweeping the grains of one kind one way, all others the other way, with one arm of the forceps or with some similar fairly dull-pointed instrument. The most abundant or the most easily recognized mineral or rock type should be separated first; then the separation of each successive type may be made in much less time than the one preceding.

The chief groups into which most Wisconsin sands fall are the following:

Translucent quartz.	Dolomite (and limestone).
Chert and feldspar.	Shales.
Quartzite, graywacke.	Granites (or all phanerites together).
Heavy minerals.	Greenstones (and slates).

Some of these are missing in many sands, and in some sands it may be found desirable to separate these still further. It is unfortunate that the separation of chert from feldspar is often difficult without the use of polarized light, as the two have very different origins. The source of the sand and the abundance of phanerites in it often show whether or not any considerable quantity of feldspar is to be expected in it.

The properties which are most reliable in a swift classification of sand grains are the following: diaphaneity, luster, form and cleavage, hardness, granularity, and color. The last-named property must be used with caution.

When the sorting is complete, each type should be put in a separate envelope, appropriately labeled, as for example:

Sd. 8 10-20 Tr. Qz.

Weighing of Separates

When the hand sorting and the heavy-solution separations are all completed, the entire batch of separates must be weighed, together with all of the unsorted portions of the various sizes, if such there be. The weight of each separate is written upon the envelope containing it. These weights are then recorded for each sample upon a preliminary analysis sheet, and each weight is computed into a percentage of the total weight of the sample. If any portion was not sorted, the percentages of the separates for that size are computed as if all had been sorted. Any appreciable errors found in the sums of weights for the sizes as

compared with those obtained before should first be looked into. In the case of the heavy-solution separates, some slight loss in manipulation is to be expected, but it should be kept under 1 per cent.

If the sum of all the heavy-solution separates of a certain density is only about 1 per cent. (or less) of the total weight of the sample, as is sometimes the case with the group of density greater than 2.9, it is not worth while to make a microscopic analysis of these separates, but they may be grouped together in the statement of analysis.

Microscopic Analysis of Heavy-Solution Separates

Since two or more minerals or rocks of different character may have the same density, and since the heavy-solution separation is subject to minor imperfections, as noted above, it is necessary to make an examination of each of the heavy-solution separates to determine its composition. A good hand lens is sufficient for this purpose, and a black sheet with a square pattern of white lines on it makes the most satisfactory background for a quantitative determination. A bacteriological counter is excellent. A representative portion of the separate is spread out evenly on a glass plate lying upon the counting sheet. If there are grains of one or several varieties scattered through a preponderance of one other mineral, as is usually the case, the former may be counted over a large area; the latter over a few of the squares included in that area, and the number multiplied by the proper factor to obtain the total. The proportions of the several minerals are reckoned to percentages, and recorded on a second preliminary analysis sheet. If the separate to be examined is a large one, more than one part of it may be subjected to counting, and an average taken. For sand of the 20-to-40-mesh type, 4 or 5 sq. cm. is a suitable area to cover for the count; for 40-to-100-mesh, 1 or 2 sq. cm. may be sufficient.

The weight of each separate having been previously computed into percentage of the total weight of the sample, the amount of each mineral (or rock type) present in the separate may be reckoned into comparable figures by means of the proportions found in the microscopic count. If the same mineral is present in more than one of the heavy-solution separates from one size of a given sample (due to inconstant specific gravity or to imperfect manipulation), the sum of these figures for that mineral in the several separates is the amount of that mineral in the size (still expressed in percentage of the total weight of the sample).

Computation of Analysis and Preparation of Report

The time included under this heading in the outline given above covers also other computations made during the progress of the analysis.

The amounts (in percentage of the total, not in grams) of each mineral and rock type found in each size are now entered on one sheet, and added to give the total amount for each mineral and rock. The analysis may then be summarized as follows, though other groups may be used if appropriate:

Igneous rocks.

Shale group.

Quartz group (includes quartz, chert, and quartzite).

Dolomite group (calcareous sediments).

Feldspar.

Heavy minerals.

Not examined (often the sand which passes 100-mesh—when less than 1.5 per cent. of the total).

The following is an example of the record of an analysis of the simplest type.

SAND 21

Mineralogical Analysis

Weighing of Sizes	Weight in Grams	Per Cent. of Sample
Plus 10 mesh.....	41.22	13.10
10 to 20 mesh.....	17.40	5.51
20 to 40 mesh.....	150.38	47.83
40 to 100 mesh.....	103.17	32.75
Minus 100 mesh.....	2.54	0.81
	314.71	100.00

Hand-Sorted Separates	Group I Coarser than 10 Mesh			Group II 10 to 20 Mesh		
	Weight in Grams	Per Cent. of Group	Per Cent. of Total Sample	Weight in Grams	Per Cent. of Group	Per Cent. of Total Sample
Igneous rocks.....	6.47	15.7	2.05	0.53	8.7	0.48
Shale group.....	4.58	11.1	1.46	1.52	24.8	1.37
Quartz group.....	10.38	25.1	3.29	2.42	39.5	2.18
Dolomite group.....	19.79	48.1	6.30	1.14	18.6	1.02
Feldspar.....	0.45	7.3	0.40
Heavy minerals.....	0.07	1.1	0.06
Total sorted.....	41.22	6.13
Not sorted	11.27
Total of group.....	41.22	100.0	13.10	17.40	100.0	5.51

Heavy-Solution Separates	Group III 20 to 40 Mesh			Group IV 40 to 100 Mesh		
	Weight in Grams	Per Cent. of Group	Per Cent. of Total Sample	Weight in Grams	Per Cent. of Group	Per Cent. of Total Sample
D > 2.9	2.26	1.5	0.72	1.03	1.0	0.33
D 2.7 to 2.9	28.74	19.1	9.13	5.36	5.2	1.70
D 2.585 to 2.7	114.12	75.9	36.31	94.10	91.2	29.87
D < 2.585	5.26	3.5	1.67	2.68	2.6	0.85
	150.38	100.0	47.83	103.17	100.0	32.75

Microscopic Counts

	20 to 40 Mesh		40 to 100 Mesh	
	Per Cent. of Class	Per Cent. of Total Sample	Per Cent. of Class	Per Cent. of Total Sample
D 2.7 to 2.9:				
Dolomite group	93.2	8.51	98.8	1.68
Igneous rocks	5.7	0.52
Quartz group	1.1	0.10	1.2	0.02
	100.0	9.13	100.0	1.70
D 2.585 to 2.7:				
Quartz group	99.2	36.02	96.9	28.94
Feldspar	0.8	0.29	3.1	0.93
	100.0	36.31	100.0	29.87
D < 2.585:				
Quartz group	14.2	0.24	17.5	0.15
Feldspar	85.8	1.43	82.5	0.70
	100.0	1.67	100.0	0.85

D > 2.9: Assumed to be all heavy minerals.

Summary

Igneous rocks	3.05
Shale group	2.83
Quartz group	70.94
Dolomite group	17.51
Feldspar	3.75
Heavy minerals	1.11
Not examined	0.81
Total	100.00

The Mellen Rod-Casting Machine

BY R. C. PATTERSON, JR., E. M., NEW YORK, N. Y.

(San Francisco Meeting, September, 1915)

IN view of the circumstance that very few important changes have been made within the last 15 or 20 years in the equipment of rod and wire mills, the description of a new process introduced by Grenville Mellen, of Llewellyn Park, N. J., to take the place of the present laborious system of producing rods of lead, zinc, brass, copper, aluminum, etc., may be interesting to members of the Institute. This new process consists in the production of cast rods at one operation in a small continuous casting machine.

The hot liquid metal is transferred from the melting crucibles directly into an endless chain of mold blocks in the machine, where solidification takes place, and the rod comes out continuously in a solid form at one end as long as the molten metal is supplied. The operation of these mold blocks so as to produce a solid rod of uniform structure constitutes an important part of Mr. Mellen's invention.

The machine (Figs. 1 and 2), 12 ft. in height over all, 2 by 3 ft. in horizontal cross-sectional area, and 6,000 lb. in weight, has a framework of cast iron, holding in position two endless chains of mold blocks, which are made in sections and join in center alignment. The mold orifice is made up of these mold sections, into which the liquid metal flows.

The molding chains are composed of steel blocks, grooved on one face to form the molding cavity, and linked together with flexible joints. Each block is carried on four rollers which guide the chain around the end sprockets and carry it in its course through the machine.

The accuracy of alignment requisite to the production of a perfect rod in the mold groove formed by these sections is secured by careful machine work, and by building the four ways, down which the alignment takes place, so that two of the sides are fixed permanently, while the opposite sides are held to their position by spring pressure. The guides carrying the molds, while in casting position, are water-cooled square tubes.

The length of this machine is somewhat indeterminate. A certain amount of both time and cooling surface is required to solidify the metal in the mold; and one may either use a long molding chain and run it fast, or a shorter chain and run it correspondingly slower. We have found,

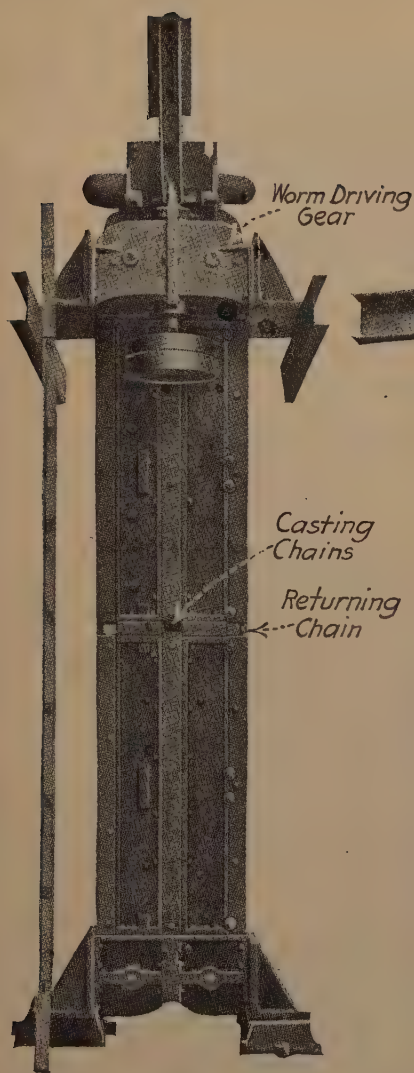


FIG. 1.

FIG. 1.—FRONT VIEW OF VERTICAL TYPE OF MELLEN ROD MACHINE. THIS MACHINE CASTS BRASS, COPPER, ZINC, ETC.

FIG. 2.—SIDE VIEW OF VERTICAL TYPE, SHOWING CASTING CHAIN WHICH TRAVELS UPWARD TO MEET IN A COMMON CENTER THE OPPOSITE CHAIN, THESE TWO UNITING TO FORM A SOLID STEEL MOLD INTO WHICH THE MOLTEN METAL IS CAST THROUGH A SILICA TUBE EMERGING FROM A GRAPHITE CRUCIBLE.

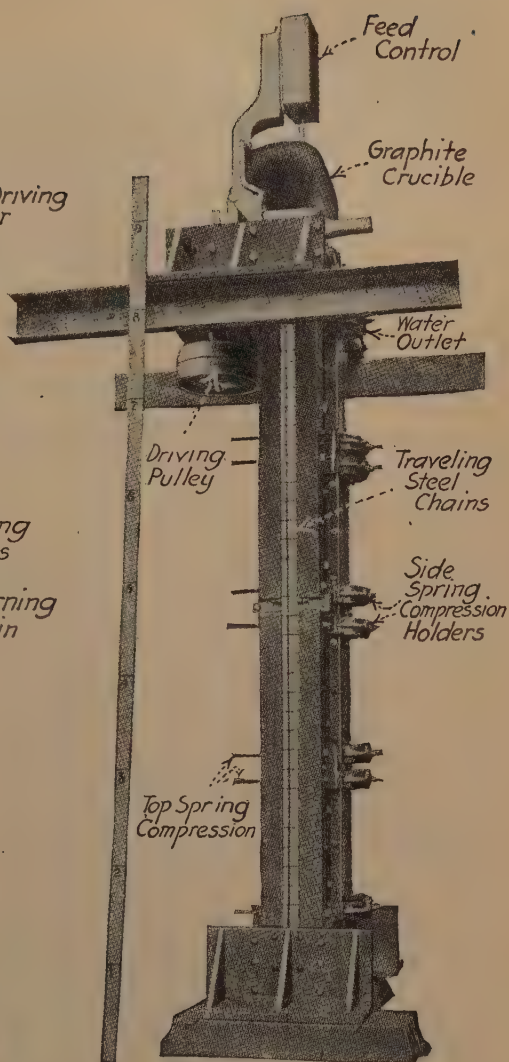


FIG. 2.

however, that with 8 ft. length between centers for the chains a casting speed of 25 ft. per minute for $\frac{7}{8}$ -in. brass rod will not cause the molding blocks to rise in temperature above 450° F.

Experiments have shown that in the casting of bronze mixtures, the pair of molding blocks should have about six times as much sectional area as the cast rod, to allow a reasonable temperature difference for the rapid removal of the heat.

The first machines designed for this process of continuous casting were vertical; but the desire of practical rod-mill men to operate with the molding groove horizontal led Mr. Mellen to spend considerable time in experimental improvements of the mechanical details, in order to secure a maximum efficiency for the horizontal machine. In this attempt, numerous troubles and intricate problems were encountered. The machine would always cast rods; but its successful operation in a horizontal position required more skill than that of an ordinary laborer. After much experiment the axis was first made slightly inclined; but finally the vertical position was readopted; and in this form one laborer, with a little preliminary instruction, can manage the machine without difficulty.

The chief hindrances to horizontal operation were: (1) the difficulty of completely filling the molds, and (2) the circumstance that, in working horizontally, the lower chain did the greater part of the work and became after a time excessively heated, while the upper chain remained relatively cool. This caused an irregular structure both in the cast rod and in the drawn product resulting therefrom; and as a result the required strength tests were not satisfactory.

In the vertical arrangement these difficulties automatically cured themselves since the metal completely filled the mold, with uniform contact all round, thus causing each portion of the chain to do its work, and giving a symmetrical structure to both rod and resulting wire.

The importance of making all of the chain work all the time is shown by the fact that in the vertical type of machine the chain can be considerably shortened, and yet the machine is capable of running at higher speed than when arranged horizontally. This vertical arrangement, however, necessitates operation on two levels.

The flow of metal into the machine is controlled by an electrically operated automatic device, which adjusts the feed to the speed at which the machine is operated. If for any reason the machine should stop the flow would be automatically shut off.

A safety device is incorporated into the drive, so that if any foreign material clogs the chain a safety pin is sheared, thus protecting the mechanism from injury.

The rods are delivered from the machine immediately to the die of the bull block, where they are drawn down to suit particular orders.

This new method of manufacturing rods does away with the following steps of the old system:

1. Casting the wire bar.
2. Handling the wire bar from the molds.

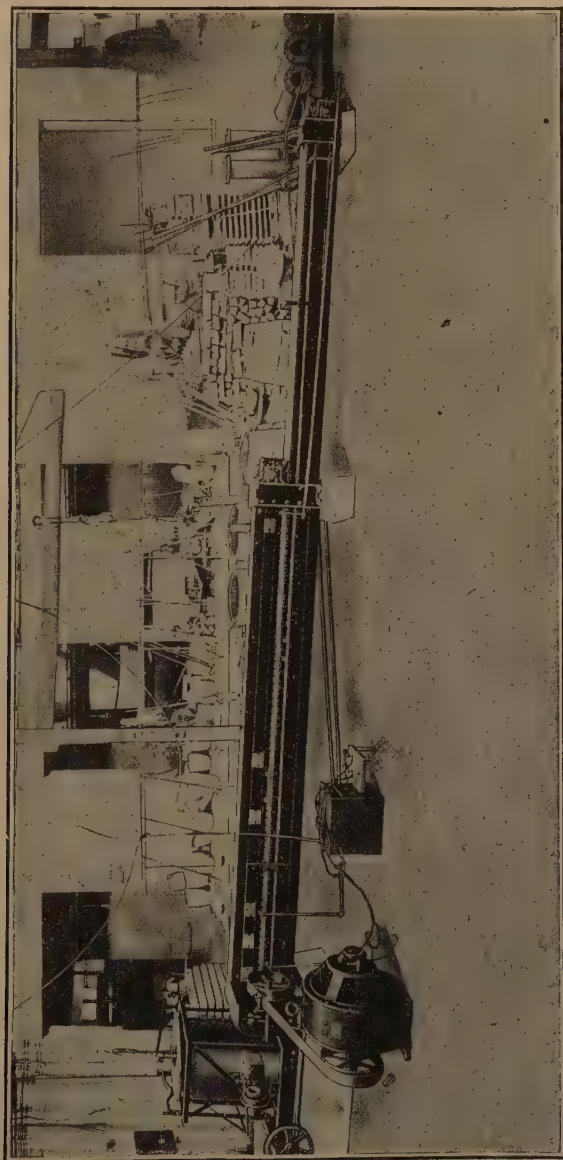


FIG. 3.—THE HORIZONTAL TYPE OF MELLEN ROD MACHINE FOR LEAD AND ALUMINUM.

3. Rehandling the wire bar to and from the reheating furnace.
4. Reheating the wire bar.
5. Rolling the wire bar.

It also eliminates the loss from oxide scaling during heating and rolling. These economies involve, of course, the saving of much capital, now invested in heating furnaces, rolling mills, power plants, land, and buildings. Since the power required to run this machine is only about 5 hp., the cost of large engines, boilers, coal bins, etc., is practically done away with.

Another advantage is the elimination of danger to human life. The apparatus is completely inclosed, so that the operator is protected from injury. Finally, the labor cost of this process is but 5 per cent. of that of the old method, the caster being the only workman required.

A later development of this continuous casting machine is the machine (Fig. 3) casting rods of lead and soft-metal alloys, $\frac{1}{4}$ in. and upward in diameter. This is the first step in the manufacture of shrapnel bullets; and the machine will cast per hour enough lead rod for more than 200,000 bullets. The only motive power required is that which is necessary for overcoming the friction resisting the motion of the traveling mold chains, which is in the neighborhood of from 2 to 3 hp.

This lead machine is practically the same as that used for brass with the exception that it is operated in a horizontal plane. Also, the horsepower is slightly decreased, and the machine is a few feet longer.

In two drawings, the area of brass rod is reduced about 27 per cent. without annealing; but it must be realized that the area reduction and pull are variables depending upon the mixture cast. An 80-hp. motor is used in handling the $1\frac{1}{4}$ -in. rod.

The continuous casting machine is also applicable to the production of aluminum and zinc rod. Aluminum rods are produced with diameters ranging from $\frac{3}{4}$ in. upward.

The casting-shop arrangement is shown by Fig. 4. This may be considered a cross-section of the shop with a long line of melting furnaces in parallel to the one shown, all of which supply metal to one casting machine. The melting furnace *A* is one of 40, the crucibles of which have a capacity of 220 lb. each with six heats per 10 hr. All of these are considered just sufficient, when operating at full capacity, to satisfy one casting machine in 10 hr. The feed basin *C* consists of a graphite crucible and has a capacity of 194 lb. The automatic feed-control device *D* consists of a solenoid magnet operating in a battery circuit, one terminal of which is grounded on the machine and the other terminal attached to a graphite point projecting into the mold orifice. When the metal in the mold rises to this graphite terminal it closes this circuit and the magnet is energized. This magnet operates upon a beam to partly close a carbon valve controlling the inlet of metal into the mold bore. When the rod that is being cast into the molds is carried below this graphite tip, the circuit is broken and the valve lifts, allowing more metal to enter the machine.

The variable-speed motor *E* is used to drive the machine. The varia-

tion in speed is necessary to enable the machine to be operated on different brass mixtures, some of which flow more freely than others. This speed is suitably adjusted by the operator at the speed-control wheel *G*.

The reeling device *F* takes the rod delivered from the machine and rolls it into bundles to facilitate its further handling. The cooling water

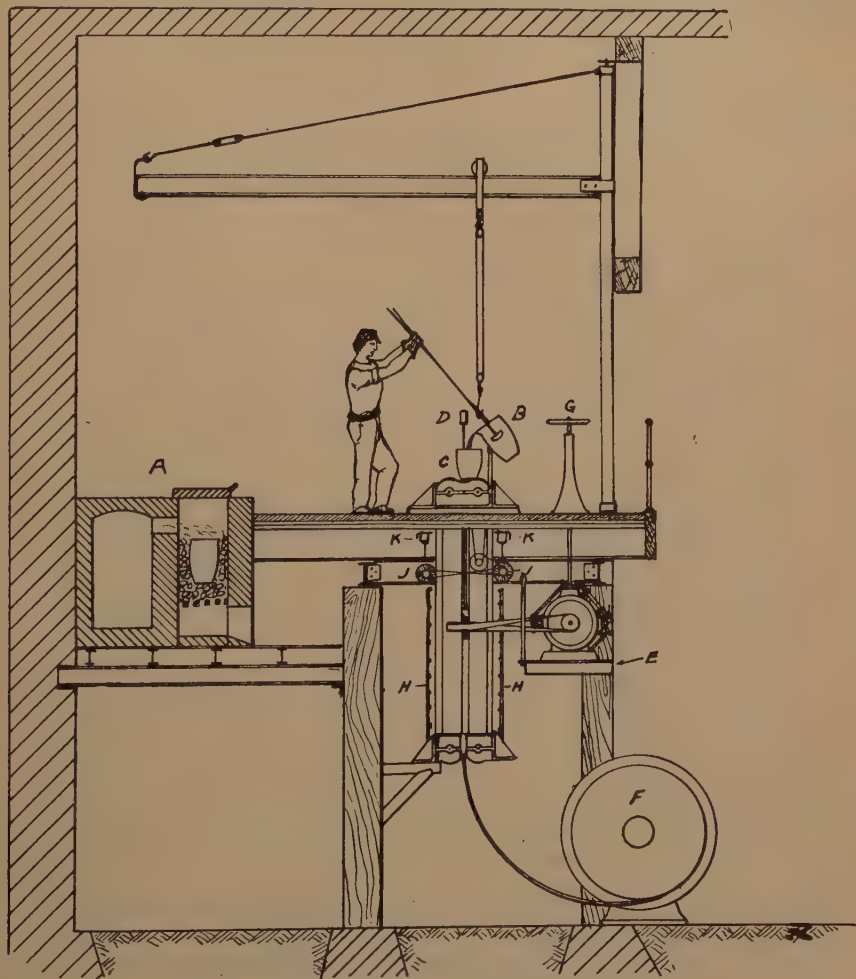


FIG. 4.—ARRANGEMENT OF CASTING SHOP.

- A. Melting Furnace
- B. Melting Crucible
- C. Feed Basin
- D. Automatic Feed Control
- E. Variable-Speed Motor

- F. Reeling Device
- G. Speed-Control Wheel
- H. Cooling Water Jets
- J. Mold-Cleaning Brushes
- K. Air-Brush Mold Facer

jets *H* are a series of pipes arranged opposite the faces of the returning molds through which a series of streams of water are played directly upon the mold faces for cooling purposes. These streams are not turned on

until the molds have reached a temperature sufficient to vaporize this water. In practice the molds appear perfectly dry a few inches above the uppermost jet. Above these water jets are noted the mold-cleaning brushes *J*. These, which are made of steel, revolve by a chain mechanism and serve to clean any deposit from the surface of the mold and prepare it to receive the necessary facing, which is applied in liquid form by means of air brushes *K*.

The machine's operating cost is less than 30c. per ton of molten metal into the form of rods.

U. S. and foreign patents have been secured for this invention.

Biographical Notice of John Birkinbine

BY ROSSITER W. RAYMOND, NEW YORK, N. Y.

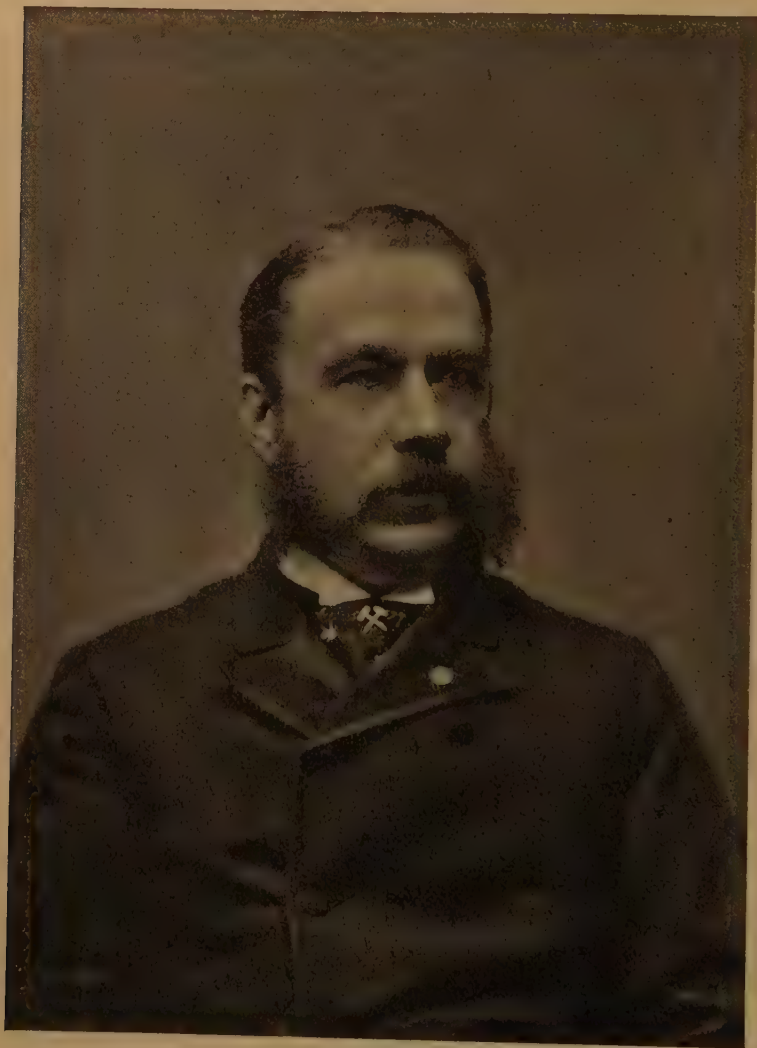
JOHN BIRKINBINE was born Nov. 16, 1844, at Reading, Pa., the eldest son of H. P. M. Birkinbine, widely known as a hydraulic engineer. The family removed subsequently to Philadelphia, where, as a young man he established with his father the office now continued by his sons, after more than 60 years.

His education was received at public schools, the Friends' High School in Philadelphia, the Hill School at Pottstown, Pa., and the Polytechnic College of Pennsylvania, where his studies were interrupted in 1863-4 by service in the Union Army, which included participation in the battles at and around Gettysburg. Later, he devoted two years to work in a machine shop; and subsequently he became associated with the late P. L. Weimer, under the firm name of Weimer & Birkinbine, operating the Weimer Machine Works at Lebanon, Pa.

Much of his work was in mining, metallurgy, and blast-furnace construction. As manager for the South Mountain Mining & Iron Co. he carried on experiments with various fuels for iron-ore smelting while maintaining the furnace in constant operation. The carefully recorded results obtained were widely published, and are referred to in text-books by other metallurgists as most complete.

From his Philadelphia office he was sent to nearly every State, and to Canada and Mexico, for examinations, reports, constructions or improvements in iron-ore mines, blast furnaces, iron works, water supplies, hydraulic development, irrigation projects, etc., and his engineering knowledge was requisitioned by several European corporations. A number of business trips were made to Mexico, beginning with a visit to the Cerro de Mercado at Durango, before railroads were established in that portion of Mexico, to make a critical examination and report on this "Iron Mountain." Later visits covered other localities and engineering problems, familiarizing him with the major part of the iron industry in Mexico. The late disturbed political conditions in that Republic have retarded the probable enlargement, modernization, and improvement of much of the iron and steel industry of Mexico, upon which he investigated and reported for various capitalists on both continents. One interesting subject included in these reports was a proposed electric furnace, to be operated by energy from water power.

Mr. Birkinbine was one of the first to suggest an iron industry at the head of the Great Lakes, using coke made from Pennsylvania coal. His report was an important factor in establishing the iron industry at the head of Lake Superior; and the blast furnace at West Duluth, Minn.,



JOHN BIRKINBINE.

was built under his plans and supervision. He was engaged by the State of Texas to investigate the practicability of iron manufacture in that State. As an engineer he co-operated with E. S. Cook of Pottstown, Pa., who did much to advance the iron blast-furnace industry. He was for

some years Consulting Engineer for the Philadelphia & Reading Coal & Iron Co., and held a similar position with Thomas A. Edison during the latter's early experiments in magnetic concentration of iron ore, and with Witherbee, Sherman & Co. in beneficiation tests; also for the Colorado Fuel & Iron Co. in the enlargement and improvement of their works and the construction of an augmented water-supply system.

In his reports and recommendations, his conclusions were clearly stated, and a reputation for conservatism and fairness brought him numerous cases of valuation, adjustment, and arbitration, in some of which, by mutual consent, he was the representative of both parties.

He always adhered to the policy of accepting no financial interest or contingent fees whatever, and would patent none of his numerous improvements or ideas, so that personal bias in his statements or conclusions could not even be suggested.

He also acted as an expert adviser for investing capitalists, and for a number of the greatest industrial corporations and several large railroad companies in this country. He was Chief Engineer, Vice-President, and Chairman of the Committee of Awards of the National Export Exposition, served on Juries of Awards at the Centennial, World's Columbian, Pan-American, and Cotton States General Expositions, and was named for similar duties at others.

From its inception in 1905 he was Chairman of the Water-Supply Commission of Pennsylvania, patriotically devoting to this work, for a nominal recompense, a large portion of his valuable time. As a result, he established an efficient organization, not only free from political influence, but noted for the zeal and faithfulness with which each member performed his duties.

He was active in forming the Pennsylvania Forestry Association, the largest and most influential of its class, and was its President for 23 years, during which time the Association accomplished the appointment of a State Forestry Commission (later made a State Department) and the enactment of statutes which encouraged the forestry movement. From the establishment of this Association, he edited its publications.

He was also active in the formation of the United States Association of Charcoal Iron Workers, of which he was Secretary, and for nine years the editor of its journal. For many years he was Special Agent for the United States Geological Survey, preparing the reports on Iron Ores for the 11th and 12th Censuses, and that on Manganese Ores for the 12th Census, and has since prepared for the Survey additional data and studies. He was appointed by the Secretary of the Interior Expert Metallurgical Engineer for the Bureau of Mines. He received marks of approval from the Survey and from several foreign scientific societies, and was member of a number of international congresses.

For 10 years he served as President of the Franklin Institute. He was

also a member of the American Society of Mechanical Engineers, the Engineers' Club of New York, the American Society for Testing Materials, the Engineers' Club of Philadelphia (President in 1893), the Manufacturers' Club of Philadelphia, the Pennsylvania Foundrymen's Association, the George G. Meade Post No. 1, G. A. R., of Philadelphia, and was an Honorary Member of the Canadian Mining Institute.

Mr. Birkinbine refused honorary degrees from two colleges, modestly saying that as he had been unable to graduate from his own *Alma Mater* he was not warranted in accepting a higher degree.

During his career Mr. Birkinbine also maintained his specialty of hydraulic engineering, acting as engineer on water supplies for various municipalities. He not only witnessed, but had active participation in, the development of water power for electrical energy. While he was at college, electricity was a laboratory experiment only, and its first exhibition as an illuminant was at Philadelphia about 12 years later; while the use of water power was then confined to limited volumes at low heads for direct mechanical purposes. His activities covered the development of hydro-electric science to its present advanced stage. In 1888 he prepared a comprehensive report on the development of the great water power of the St. Louis River in Minnesota, considering a 15-mile transmission, though no water-wheel manufacturer would guarantee turbines for heads above 35 feet. Since then he was associated with or reported on many developments in various States and in Mexico, covering high heads or large volumes of water until lately deemed impracticable.

Mr. Birkinbine became a member of this Institute in 1875, a Manager in 1883, Vice-President in 1887, and President in 1891 and 1892. The following is a list of his contributions to the *Transactions*:

Papers

Title	Vol.	Page	Year
Suspended Hot-Blast Stoves.....	IV	208	1875
Pumping Engines.....	V	455	1876
Notes upon the Drainage of a Flooded Ore-Pit at Pine Grove Furnace, Pa.....	VI	174	1878
The Production of Charcoal for Iron Works.....	VII	149	1878
Experiments with Charcoal, Coke and Anthracite in the Pine Grove Furnace, Pa.....	VIII	168	1879
A Short Blast at the Warwick Furnace, Pennsylvania.....	IX	51	1880
Charcoal as a Fuel for Metallurgical Processes.....	XI	78	1882
Roasting Iron-Ores.....	XII	361	1883
The Cerro de Mercado (Iron Mountain) at Durango, Mexico.....	XIII	189	1884
The Distribution and Proportions of American Blast-Furnaces.....	XIV	561	1885
Operation of Warwick Furnace, Pennsylvania, from August 27th, 1880, to September 1st, 1885.....	XIV	833	1886

Papers.—Continued

Title	Vol.	Page	Year
Comparisons of Blast-Furnace Records.....	XV	147	1886
A Tilting-Ladle Car for Molten Metal or Slag.....	XV	685	1887
The Distribution and Proportions of American Blast-Furnaces. (Second Paper.).....	XV	690	1887
The Resources of the Lake Superior Region.....	XVI	168	1887
Prominent Sources of Iron-Ore Supply....	XVII	715	1889
Crystalline Magnetite in the Port Henry, New York, Mines.....	XVIII	747	1890
Progress in Magnetic Concentration of Iron-Ore.....	XIX	656	1890
The Fuel-Supply of the United States. A Sketch of the Progress of Twenty Years in the Economy of Production and Consumption. (Presidential Address.).....	XX	409	1891
The Influence of Location upon the Pig-Iron Industry. (Presidential Address.).....	XXI	473	1892
Industries of the Schuylkill Valley. (Presidential Address.)	XXI	618	1892
The Development of Technical Societies. (Presidential Address.).....	XXI	962	1893
Note on a Supposed Aztec Mirror.....	XXIV	617	1894
Note on a Piece of Carpenter Steel.....	XXIV	619	1894
The Iron-Ore Supply....	XXVII	519	1897
Distribution of the World's Production of Pig-Iron.....	XXX	504	1900
Hydraulic Pumping-Plant on the Snake River, Idaho, for Power, Irrigation and the Treatment of Gold-Sands....	XXX	518	1900
Growth of the Pig-Iron Production During the Past Thirty Years. (Not Printed.).....	XXXIII	xxxvi	1902
Biographical Notice of William George Neilson.....	XXXVIII	402	1907
The American Institute of Mining Engineers and the Conservation of Natural Resources.....	XL	412	1909

Remarks in Discussion

Blast-Furnace Hearths and In-Walls.....	IV	186	1875
The Economy Effected by the Use of Red Charcoal.....	VI	204	1878
Experiments with Straight or No-Bosh Blast-Furnace....	XIII	498	1884
An Improved Langen Charger.....	XIII	528	1884
Comparison of Some Southern Cokes and Iron-Ores....	XV	754	1887
Development of American Blast-Furnaces.....	XIX	992	1890
The Iron-Mining Industry of New Jersey.....	XX	224	1891
American Blast-Furnace Practice.....	XX	266	1891
Manganese in Cast-Iron.....	XX	315	1891
The Magnetic Concentration of Iron-Ore.....	XX	595	1891
Preservation of Hearth and Bosh-Walls.....	XXI	118	1892
The Magnetic Iron-Ores of Ashe County, N. C.....	XXI	277	1892
Discussion on the Crushing of Iron Ore.....	XXI	548	1892
The Hugh Kennedy Hot-Blast Stove.....	XXI	735	1892
Nickel and Nickel-Steel.....	XXV	961	1895
Removal of Sand from Waste-Water of Ore-Washers....	XXVIII	842	1898
Important Results Obtained in the Past Fifteen Years with the Stiff and Heavy Rail-Sections.....	XXIX	1015	1899
Blast-Furnace Practice.....	XXXVI	796	1905

Mr. Birkinbine always maintained a friendly interest in his fellow members of the profession, and held to the thought that engineers were coöperators and not competitors. He continued his personal interest in all associates and was ever ready to help young men by advice.

As a citizen he was always active in promoting the public good. He served on the Civil Service and other Commissions, as well as rendering professional services for the City of Philadelphia. His neighborly activities were maintained up to the end of his life. His last days were happily spent among his family, for as a devoted and Christian husband and father he fulfilled what he considered his greatest pleasure and the noblest of all his duties.

He died May 14, 1915, at his home in Cynwyd, near Philadelphia. Simple funeral services and private burial were held in accordance with his known desire; but the former were attended by representatives of many organizations and by professional colleagues and personal friends from many places.

Kick vs. Rittinger: An Experimental Investigation in Rock Crushing, Performed at Purdue University

BY ARTHUR O. GATES,* B.S., M.E., MISHAWAKA, IND.

(San Francisco Meeting, September, 1915)

INTRODUCTION

RITTINGER's law of the energy expended in crushing is, as roughly stated by Professor Richards,¹ that the work of crushing is proportional to the reduction in diameter; or, as I have more fully expressed it:

"The work done in crushing is proportional to the surface exposed by the operation; or, better expressed for this purpose, the work done on a given mass of rock is proportional to the reciprocal of the diameter of the final product—assuming that all the mass has been reduced to one exact size, which is only theoretically possible."²

Kick's law is stated by H. Stadler as follows:³

"The energy required for producing analogous changes of configuration of geometrically similar bodies of equal technological state varies as the volumes or weights of these bodies."

In other words, the energy expended is proportional to the volume reduction, instead of the diameter reduction.

That these two laws would give widely different results may be shown by a simple imaginary case. A ton of 16-in. cubes is broken to 1-in. cubes at the first operation, and these are broken to $\frac{1}{16}$ -in. cubes at a second operation. Since the first operation produces only one-sixteenth as much new surface as the second, the ratio of energy expended in the two operations would be by Rittinger's law as 1 to 16. By Kick's law, since the volume-reduction ratio is the same in both cases, the ratio of energy would be 1 to 1. This discrepancy is great enough to challenge a test by actual experiment, of which there has been, hitherto, but little compared with the amount of argument. The present paper is the record

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¹ *Ore Dressing*, p. 304 (1903).

² *Engineering and Mining Journal*, vol. xcv, No. 21, p. 1039 (May 24, 1913).

³ *Grading Analyses and Their Application, Transactions of the Institution of Mining and Metallurgy*, vol. xix, p. 478 (1910-11). The original statement will be found in *Das Gesetz der proportionalen Widerstände*, by F. Kick.

of such a test, and is offered as showing that Rittinger's theory more nearly represents the actual facts than any other proposed hitherto.

Mr. Stadler, in the paper already cited, applying to rock fracture Kick's law (which I accept as applicable to the deformation of elastic bodies), proves that the stamp mill, working on relatively coarse feed, is several times as efficient as the tube mill on fine feed, this relation being expressed in what he calls "relative mechanical energy per horsepower." He distributes his energy units among the various sizes produced, according to Kick's law. But my results show, among other things, that many more of Stadler's energy units are obtained per foot-pound-applied in coarse than in fine crushing. This suggests a doubt as to the correctness of his unit, since one would expect a foot-pound-applied to produce the same number of "energy units," whether the feed and resulting product were coarse or fine. When Rittinger's theory is applied, the number of mesh-tons (my unit of surface produced) is nearly proportional to the foot-pounds-applied, whether the product be coarse or fine; hence my conclusions favor the simple Rittinger theory that surface produced is proportional to energy applied.

THE TWO LAWS

1. It is the purpose of this paper to record and interpret a series of experiments made for the purpose of studying the consumption of energy in the crushing or fracturing of rock particles both coarse and fine. At the start, the expectation was to determine constants according to the Rittinger theory for several rocks, whereby it would be possible to calculate absolute crushing efficiencies of machines crushing such rocks, or to predetermine what a machine would accomplish in the way of tonnage and screen analysis upon a given rock, knowing what the machine accomplished upon another rock whose constants were likewise known. While approximate constants for a few rocks were determined, the results indicated that an amount of work (perhaps with the microscope upon sizes too small to be screened) greater than could be accomplished by a single investigator would be necessary to determine these constants with reasonable accuracy. For this reason no constants are submitted as such in this paper.

2. It is possible, however, to compare by means of these experiments the two antagonistic "laws" of crushing already mentioned. The fracture of cubes by compression in a testing machine produces, in nearly all cases, pyramids of the same general shape (Fig. 1), indicating that the stresses are greatest along certain planes of fracture; and, since the relative movement must have been greatest along these fracture or shear planes, the greatest amount of energy must have been expended there. While I know of no tests that show that the several pieces return to the

dimensions existing before the break, it seems reasonable that they do, and that they do not retain the deformation. All that has resulted from the pressure and consequent movement is *cracks*. They are new, while the particles resulting are of the same form and specific gravity as they were before separation, and if put together accurately the dimensions of the mass would be the same.

Rittinger's Law

3. Rittinger's theory is the law of the *cracks*. It is well explained in an article by E. A. Hersam in the *Mining and Scientific Press*, an abstract of which is given in the third volume of Richard's *Ore Dressing*, and to which I acknowledge my indebtedness for some of my ideas and methods.

4. The mathematics of this theory follow:

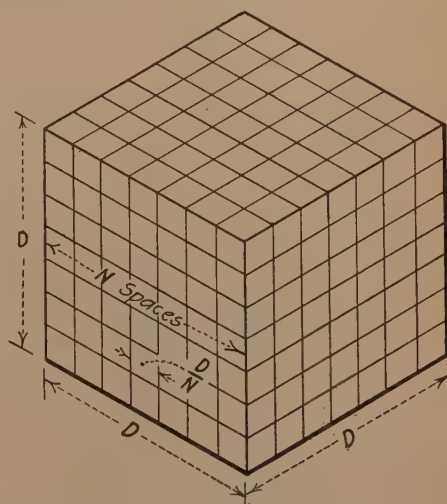
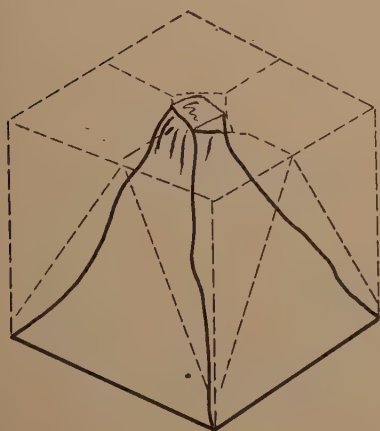


FIG. 1.—FRACTURE OF CUBE IN PRACTICE. FIG. 2.—THEORETICAL FRACTURE.

Assume a cube of rock (Fig. 2) of side D , divided into smaller cubes by planes passing through it in three directions, as shown. The surface of the original cube being $6D^2$, the aggregate surface of the smaller cubes will be $6ND^2$, and the new surface exposed will be $6(N-1)D^2$. The theoretical mesh or reciprocal of diameter of new particles, M , is $\frac{N}{D}$, whence $N = MD$; and the aggregate final surface, $S = 6ND^2 = 6MD^3$.

With D constant, that is, starting from the same size of cubes in both cases, $S : S' :: M : M'$, or the total surfaces of equal volumes of rock, each composed of particles of uniform size and shape, are to each other as the theoretical mesh or the reciprocal of the diameter of the particles composing the respective masses.

Suppose the surface S to be made up of x_1 parts (by weight) of M_1

mesh (theoretical), x_2 of M_2 , x_3 of M_3 , etc. x_n of M_n . Then,

$$S = k \sum_1^n xM;$$

that is, the summation of the weights by the theoretical meshes, where k is a constant, six times the volume of unit weight of the rock. If different rocks were considered, specific gravity would enter into the value of k .

Likewise,

$$S' = k \sum_1^n x'M'$$

Now where S was the surface existing on the mass before breaking, and where S' is the total surface existing after the breaking, the new surface

$$S' - S = k \sum_1^n (x'M' - xM)$$

For a given screen analysis, weights of the different theoretical meshes, we may plot on co-ordinate paper x against M and x' against M' , and obtain a crushing-surface diagram (Fig. 3) representing the equation

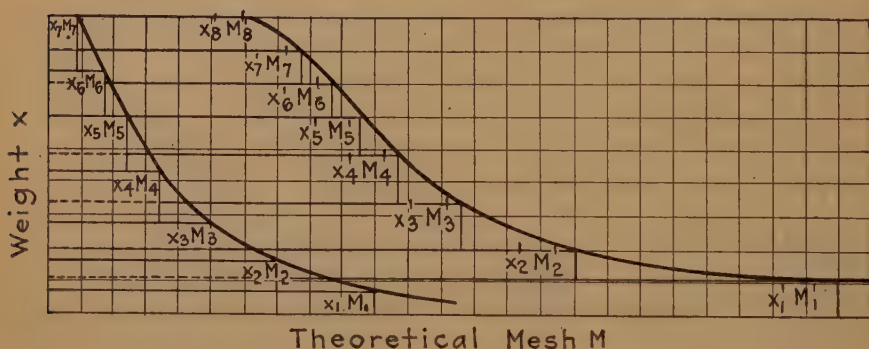


FIG. 3.—CRUSHING-SURFACE DIAGRAM.

above given for new surface, the area between the steps being proportional to the difference in surface area. If we take closer divisions of M , smaller differences in x will result, and the limits of such divisions will result in the smooth curves, the area between which is a true measure of the surface produced.

Thus, on the Rittinger theory that surface produced is proportional to energy applied, the area of the diagram is a measure of the energy going into the crushing. (It should be distinctly understood that bearing- and gearing-friction losses should not be included in the energy-applied measured by the crushing-surface diagram, and that it is therefore impossible to compare the two laws by tests upon commercial machines, unless the friction losses are known.)

THE CRUSHING-SURFACE DIAGRAM

The crushing-surface diagram has the advantage over any other graphic method of comparing screen analyses, that it shows at a glance

the "classification" of the pulp, the distribution of the surface, and (according to the Rittinger theory) the energy spent in its production. As the cards of a steam engine enable us to determine the power developed, and to locate faulty operation or design, so the crushing-surface diagram enables us to determine the power consumed, and to locate points at which the crushing practice may be improved and faults remedied in design.⁴

5. From my original article on The Crushing-Surface Diagram,⁵ I take Fig. 4 and the following theoretical proof of the Rittinger theory:

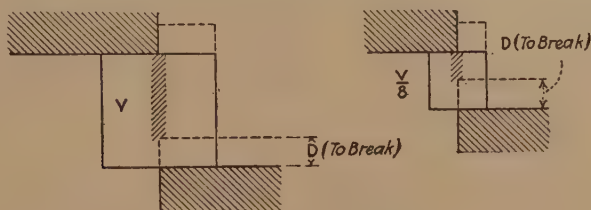


FIG. 4.—RITTINGER'S LAW.

100-Mesh Cube	200-Mesh Cube
Area one section = A	Area one section = $\frac{A}{4}$
Average resistance to shear per sq. in. = F	Average resistance to shear per sq. in. = F
Energy = FAD	Energy = $F\frac{A}{4}D$
Surface produced = $2A$	Surface produced = $2\frac{A}{4}$
\therefore Energy Proportional to Surface.	

Suppose it were possible to hold two similar cubes between the two offset faces as shown, and that forces were applied until the deformation shown by the dotted lines was obtained, it will be seen that only the molecules along the vertical center line [it should read plane] are stressed and deformed, the mass of the cube away from this surface receiving practically no pressure or deformation. The energy in this case required to produce rupture will be the product of the average resistance to shearing per square inch by the area along which rupture takes place, and by the distance the two offset faces move together. The average resistance to shearing is a variable quantity, as the deformation increases up to rupture. To reduce to cubes, this amount of energy must be multiplied by three, as three similar fracture planes must be made to produce cubes. And it will not be hard to see that the distance through which the offset faces must move in either case must be the same and not proportional to the thickness of the piece. To break the molecular bond between adjacent particles would require the same movement, regardless of the thickness of the piece.

"It will be noted that if eight of the half-diameter cubes be sheared to produce cubes of half their size, the new surface presented will be double that formed when the single large cube is sheared into half-size cubes, and also that the energy required in the case of the smaller cubes is double that required in the case of the larger cubes. This should demonstrate that energy applied to crushing is proportional to the surface produced."

⁴ This matter has been discussed in my article, Applications of the Crushing-Surface Diagram, *Engineering and Mining Journal*, Apr. 18, 1914.

⁵ *Engineering and Mining Journal*, May 24, 1913.

Of course the above passage states a theoretical case of shear, and it is not to be expected that cubes can be held in the manner shown; in practice the breaks occur, perhaps, as in Figs. 1 and 5

6. New units are apparently necessary when using the crushing-surface diagram. The term "mesh-grams" as a proposed unit for crushing constants and measurement of surface I have defined as "the surface produced by a theoretical crushing operation in which one gram of particles of the same diameter are all reduced to a diameter whose reciprocal is one greater than before reduction." The unit "mesh-ton" I have similarly defined as "the increased surface produced by crushing

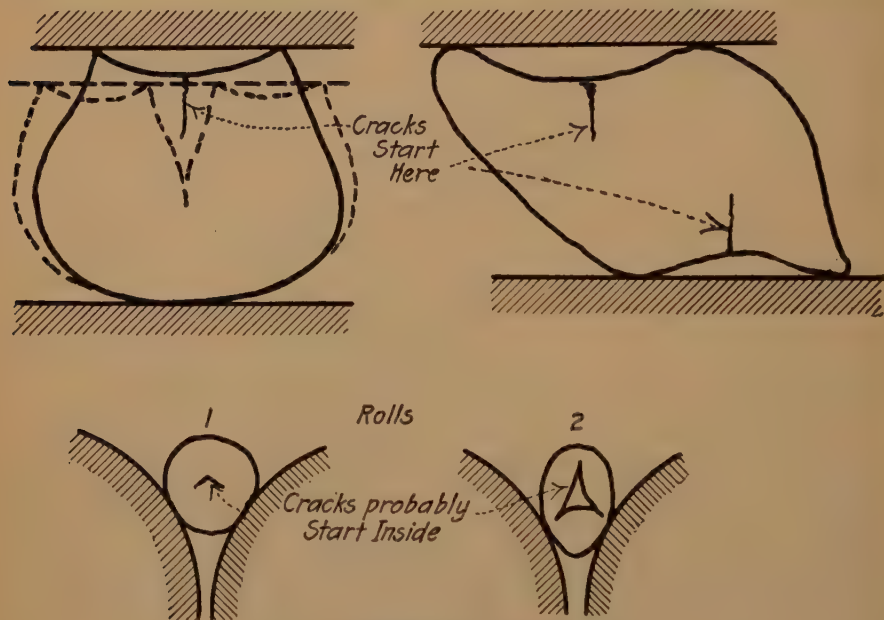


FIG. 5.—BEGINNINGS OF FRACTURE.

all particles of a ton of rock to a diameter whose reciprocal is one greater than in its previous condition."⁶

As the crushing contemplated in the definitions is only theoretically possible, a couple of examples may explain them better. Shapes remaining the same, preferably cubes for argument's sake, during all stages of the crushing, if 1 g. of 0.1-in. rock (reciprocal, 10) be crushed until it is all 0.05-in. (reciprocal, 20), the new surface produced is 10 mesh-grams. Or if 10 g. be reduced to exactly 0.01-in. from exactly 0.1-in. particles, there will be an increase of 10 times 100 minus 10 times 10, or 900 mesh-grams. Mesh-pounds, mesh-ounces, mesh-percentages, etc., are similarly justified and defined.

⁶ *Trans.*, xlvii, 59 (1913).

7. The term "mesh" for a measurement has been objected to, as ambiguous in meaning. Some other short term may be desirable to take its place in forming these compound names for surface produced. But nothing better has been suggested. "Reciprocal" (of diameter) is too general and too long; "theoretical mesh-tons" is likewise too long and is also liable to misinterpretation; "dia-ton" has been suggested; but it would hardly do to speak of "dia-grams" in connection with a diagram. Perhaps "pitch-grams," "pitch-tons," etc., would be satisfactory, pitch being defined as the theoretical mesh or reciprocal of diameter (probably in inches). I would here again point out that these quantities are measures of surface, and that the weight element is only permitted to enter on the basis of equal specific gravity; where the specific gravity varies, a correction must be made for it.

Kick's Law

8. Again I quote from my original paper my interpretation of Kick's law, Stadler's statement of which I have given above:

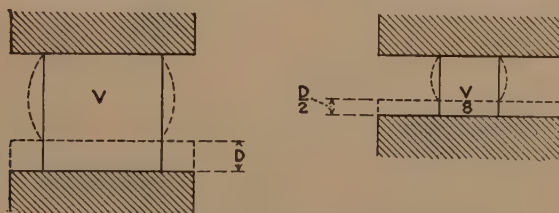


FIG. 6.—KICK'S LAW.

100-Mesh Cube	200-Mesh Cube
Area one face = A	Area one face = $\frac{A}{4}$
	Average resisting force per sq. in. = F
Energy = FAD	Energy = $F \frac{A}{4} \frac{D}{2} = \frac{FAD}{8}$
\therefore Energy Proportional to Volume.	

"This law does not apply so much to crushing as to deformation of bodies before rupture takes place.

"In Fig. 6 are represented two particles of ore 'of equal technological state,' shown as cubes between the faces of a crushing or testing machine. Assuming the theoretical mesh, equivalent to the reciprocal of diameter, and using concrete values, we have a 100-mesh particle with eight times the volume of the 200-mesh particle, and with an area per face four times that of the 200-mesh particle. The dimensions are as two to one, and the bodies being similarly deformed within the elastic limit without fracture, the energy that must be applied in each case to produce this deformation is the product of the average resisting force per square inch, the same in both cases, by the area worked against and by the distance through which this average force works. As shown in the figure, in this particular instance the energy absorbed is proportional to volume, and it can be similarly shown for the general case. On the gradual release of the external pressure the energy absorbed is given back to the machine producing the deformation [better, which produced the deformation] and the body returns to

its original shape. It should be noted that the body has been deformed only by a gradually increasing pressure, the first increment of deformation not requiring so much pressure as the last. *In case the body has been deformed beyond its elastic limit either the whole mass of particles have been reduced to the molecular state by the freeing of their bonds with adjacent particles, which never happens, or fracture takes place along a few surfaces by the breaking down of some of the weaker bonds, and the particles thus formed are free to resume their original shape in so far as they are not held between the machine surfaces.* The energy given up by them is probably used in some sort of lever action in making fracture planes. So the energy absorbed according to Kick's law does not stay in the particles after pressure is released and therefore this does not govern to any great extent the amount of energy absorbed in crushing."

It is evident to the engineer that Kick's law as explained above is but another form of the law of elastic bodies: namely, that within the "elastic limit" stress is proportional to deformation.

10. There is no incompatibility between the Rittinger theory and the laws of elastic bodies such as glass, steel, rock, etc. Up to the elastic limit, deformation is proportional to stress, and absorbed energy is

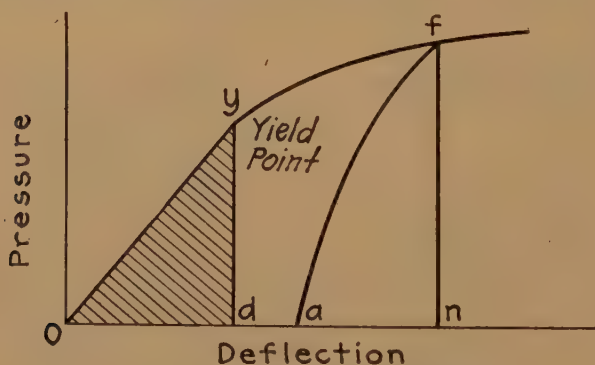


FIG. 7.—STRESS-STRAIN DIAGRAM.

proportional to the volume of the particle stressed, according to Kick's law. In Fig. 7 the area Oyd is a measure of this energy. When pressure within the elastic limit is released the body returns to its former shape, there having been no interior breaks so far. When, however, the elastic limit has been exceeded, certain local ruptures have taken place within the body at points weaker than others; these breaks occur consecutively or simultaneously at various places within the body, and upon release of pressure the body is found upon careful measurement to have taken on a permanent "set." The first fracture noticeable occurs after several ruptures have weakened the body at one point, which may be along a plane of greatest resultant stress, or of structural weakness. Referring back to Fig. 7, the elastic limit having been exceeded, as pressure is gradually released, its relation to the deformation is shown by the line af, the area afn being a measure of the energy given back to the machine

(by Kick's law), while the area *O_fa* is a measure of the energy absorbed by the local breaks, which may or may not have become visible. In the case of a wrought-iron or mild-steel test bar undergoing tension between the jaws of a testing machine, local ruptures are probably occurring throughout the bar, reducing the section at those points of weakness, and putting greater load upon adjacent particles which may be then able to carry the load better.

The final rupture of the steel bar occurs when a number of local breaks at one particular section have so weakened the bar that all the work the machine is doing is concentrated at that section, as shown by the "necking in" just before failure takes place. Unquestionably in spite of the permanent set in the test specimen away from the point of failure, the majority of the crystals composing the specimen are in the

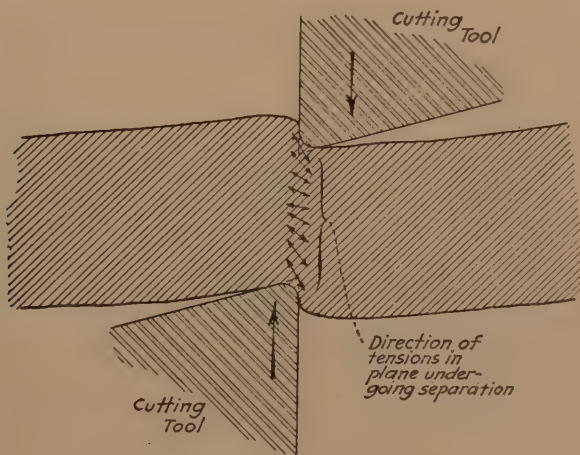


FIG. 8.—SHEARING METAL BAR IN COMMERCIAL MACHINE.

same form as before the test, but some have slipped away from others and there is a new adjustment due to the failure of the few which could not hold on, the others at that section being in position to support the load for a further period. If it were wrought iron, we could say the fiber slipped along the slag or impurities between fibers until the slipping fibers were crowded so that they obtained a sort of "strangle hold."

10. In the case of the shearing of iron and steel bars, it has been shown⁷ that the depth of penetration (the distance traversed by the cutter before separation takes place) is not proportional to the thickness of the metal, as it would be in case Kick's law applied, but is proportionally greater in thin sheets than thick—about 75 per cent. of the thickness in the case of sheets $\frac{1}{8}$ in. thick, and about 25 per cent. in the case of sheets 1 in. thick.

⁷ *American Machinist*, 1905; see references in J. D. Hoffman's *Machine Design*.

On the face of it, this fact does not uphold the theoretical argument for Rittinger any more than for Kick, but let us look further.

In a commercial shearing machine we do not get true shear, but a condition shown in Fig. 8. The material is compressed under the blades of the shear, which are themselves somewhat distorted during the operation, and failure is due to tension in a line inclined to the plane of the cutting edges. In the case of a thick plate there would be more deformation of the cutting edges due to the higher resistance of the material, and more of a width concerned than in the case of thin sheets. When a hole is punched in steel, there is a volume of material around it that must be removed for first-class work; this volume is probably full of minute fractures the area of which

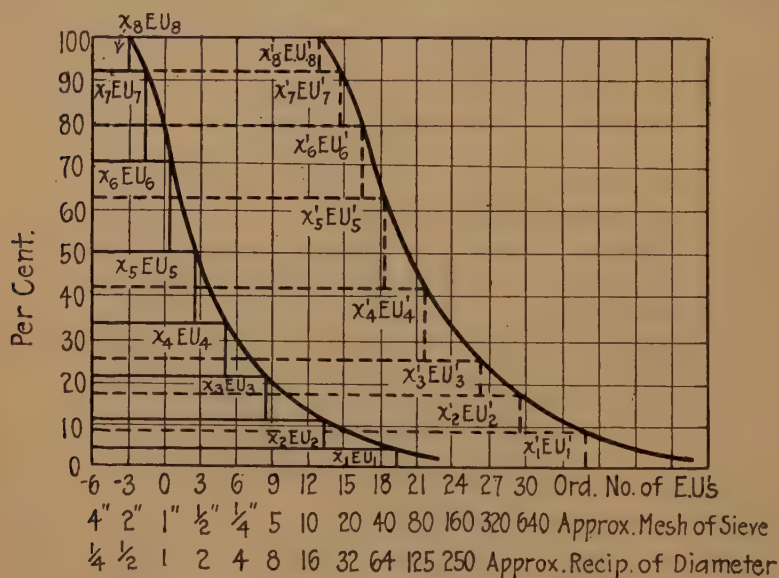


FIG. 9.—MECHANICAL VALUE DIAGRAM.

cannot be measured. I am told that rock crushed for concrete is seen, upon microscopic examination, to be full of minute fractures, due to the crushing, which cause each piece to be weaker than a natural pebble of the same mineral and size. If the rock were uniformly full of these invisible fractures, Kick's law would apply in spite of all our theoretical reasoning. Just as in the case of the hole punched in steel, I believe that these minute fractures in the rock are close to the fracture planes.

11. A. F. Taggart, has discussed⁸ Kick's law thoroughly from the standpoint of its advocates and there is no reason for a repetition of his work. However, taking it at their valuation, and using data obtained from Stadler's papers already referred to, we can by reasoning similar to

⁸ The Work of Crushing, *Trans.* xlviii, 153 (1914).

that already employed develop a diagram for energy absorbed similar to the crushing-surface diagram. (Fig. 9.)

With constant weight and similar shapes, Stadler allows one energy unit (E. U.) for each reduction to one-half the volume immediately preceding. Expressed mathematically,

$$\left(\frac{D}{d}\right)^3 = \frac{V}{v} = 2(\text{E. U.})$$

Or, as more frequently expressed,

$$\log V - \log v = (\text{E. U.}) \log 2$$

Whence

$$(\text{E. U.}) = \frac{(\log V - \log v)}{\log 2} = 3 \frac{(\log D - \log d)}{\log 2}$$

D and d being initial and final dimensions, V and v being initial and final volumes. For example: Reducing 1-in. cubes to one-half volume, 1 E. U. has been expended; reducing to one-fourth volume, 2 E. U.; to one-eighth volume, 3 E. U.; to one-fourth dimensions, 6 E. U.; to eighth dimensions, 9 E. U.; and so on. Reducing from 100 in. diameter to 1 in. the same number of E. U. are expended as from 1 in. to 0.01 in., or as from 0.01 in. to 0.0001 in. (all on quantities of the same volume, both here and in the previous sentence). Of course various sizes are produced in crushing, and each grade or size has a different value, the same reasoning applying as in section 4, above.

Suppose mechanical value W be made up of x_1 per cent. of E. U.₁, x_2 per cent. of E. U.₂, etc. . . . x_n per cent. of E. U._n; then

$$W = c \sum_1^n x \text{ E. U.}, \text{ before the crushing operation, and}$$

$$W' = c \sum_1^n x' \text{ E. U.}', \text{ after crushing.}$$

$$\text{The new mechanical value, } W - W' = c \sum_1^n (x \text{ E. U.} - x' \text{ E. U.}').$$

Plotting x against E. U., we have, when the limiting curves are drawn, a diagram (see Fig. 9) similar to the crushing-surface diagram, the area of which is a measure of the mechanical value of the crushing operation or the energy units (E. U.) absorbed. (Percentages should be expressed decimally in the above calculations.) The mechanical value of sands given in E. U. from the diagram is multiplied by the tonnage per day (or hour) and divided by the horsepower, giving the *relative mechanical efficiency*.

12. The E. U. per foot-pound applied upon 100 g. as found from these mechanical-value diagrams compared with mesh-grams per foot-pound as found from crushing-surface diagrams for the same crushing operation will, if covering a range of operations, show which of the two laws under consideration is consistent with the facts. This is the method used in

comparing my results. (In Fig. 9, it will be noted that the theoretical mesh or reciprocals of diameter are designated at the corresponding "Ordinal numbers of energy units," which is self-explanatory, and perhaps permits easier comparison with the crushing-surface diagram for the same crushing operation.) The following comparisons based upon the experimental work performed show that the Rittinger theory applies to crushing operations and that Kick's does not. There are discrepancies when we apply the Rittinger theory; but these can be explained reasonably by considering the expenditure of energy upon the sizes finer than our screen permits our measuring, which must present considerable

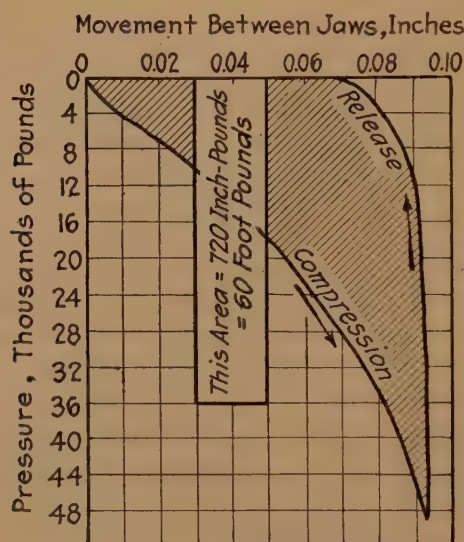


FIG. 10.—ENERGY DIAGRAM.

surface, if we assume that the curve of the screen analysis continues to follow the same law into the finer sizes as among those already measured.

METHOD OF TESTS

13. In this work one or many pieces of rock were compressed between the faces of the Amsler-Laffon hydraulic testing machine in the Laboratory for Testing Materials of Purdue University. (This machine is described in the appendix.) The movement between the faces (similar to those of a crusher) was measured by a deflectometer having a vernier on which movements of 0.001 in. could be read, while the pressure between faces was read upon a dial to a maximum of 150,000 lb. The product of the average pressure (or rock resistance) by the distance through which that pressure was exerted is work or energy, and easily reduced to foot-pounds.

I found it easier to plot my results as obtained, pressure against distance, as shown in Fig 10, the area of the closed curve, completed by the record of pressure and distance when pressure was gradually released, being a measure of the energy in inch-pounds (divided by 12 equals foot-pounds). This energy as foot-pounds, or as horsepower-hours, can be compared with the surface produced as shown by the crushing-surface diagram of the screen analyses of initial and final products of the operation; and it can likewise be compared with the energy units (E. U.) of Stadler, as shown by mechanical-value diagrams similar to Fig. 9. By crushing the same amount of rock from coarse to medium, from medium to fine, from fine to very fine, etc. (coarse and fine being merely relative terms), the two laws may be compared through a sufficient range to detect any abnormal results or inconsistencies.

14. To each test, whether it involved one or several consecutive crushing operations, was given a letter by which it is designated throughout this paper. My records are in the form of crushing-surface diagrams with the weights marked to place as plotted. From these the mechanical-value diagrams must necessarily be derived, as this work was not undertaken primarily to settle the case of Kick vs. Rittinger. The pressure-deformation curves were plotted directly in a similar manner without recording numerical readings. The advantage of recording by plotting the observations directly upon co-ordinate paper is the speed at which it is done, and the check afforded by the shape of the curve. It is hoped that the work will be acceptable in its present form, as numerical tabulations would add nothing of value to this paper. So far as possible the results of calculations are given in the diagrams, and hence it will be unnecessary for the reader to refer back and forth from text to diagram to any great extent.

15. As the crushing-surface diagram does not depend upon any definite system of screens so long as the aperture be known, all the screens used were measured with a Bausch & Lomb microscope having a scale reading to $\frac{1}{1500}$ in., and the reciprocals of diameter of these apertures were used in plotting.⁹

Certain of the finer sizes were screened through bolting cloth held in embroidery frames. I have suggested specially prepared plotting paper, *i.e.*, with vertical lines drawn in at the reciprocals of diameter corresponding to the apertures of the screens used, for such cases as the Tyler standard screen-scale sieves, or the I. M. M. standard. Since this paper was first submitted to the Institute and of course since the work was done, I have discovered that cumulative direct diagram sheets with Tyler S. S. S.

⁹ In some of the early work actual screen meshes were recorded, but for the sake to uniformity these have been reduced to reciprocals and shown as though recorded that way.

sieves are desirable for this work and that there is no need of any other sheets for crushing-surface diagrams. Fig. 11 shows a crushing-surface diagram on this paper, the plotting running in the opposite direction from that for which it was designed; 1 sq. in. on the regular sheet, with screen sizes located as indicated, representing 8.85 theoretical mesh-tons of surface per ton of ore crushed.

16. All results have been modified to a basis of 100 g., so that percentages required for use with E.U. are readily obtained, about one-third of the tests having been made upon a definite 100 g. In none of the

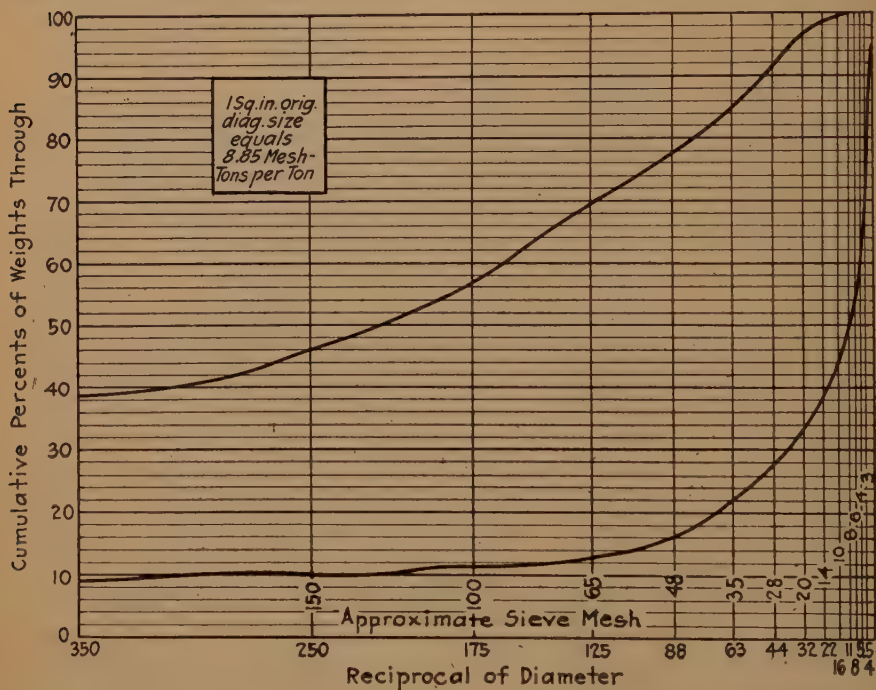


FIG. 11.—CRUSHING-SURFACE DIAGRAM ON TYLER CUMULATIVE DIRECT SHEET.

tests was any attempt made to account for the losses occasioned by fine dust floating away, or by coarser pieces shooting off as they were fractured. In no case could these be more than 2 per cent., which would not materially affect the results. Extreme accuracy was not particularly necessary in this work. It was rather a reconnaissance; and if more than three significant figures are given at any point, too much importance should not be attached to the last figure. At the same time the work was carefully done and the record carefully made, but there is so much unknown that I would not care to defend the fine details of my results.

RECORD OF TESTS

17. In Fig. 12, A and B are results obtained upon a tube-mill feed, a fairly siliceous gold and silver ore from Mexico, of which 100 g. of $-\frac{5}{16}$ -in. +4-mesh pieces were crushed by two movements of the machine, without

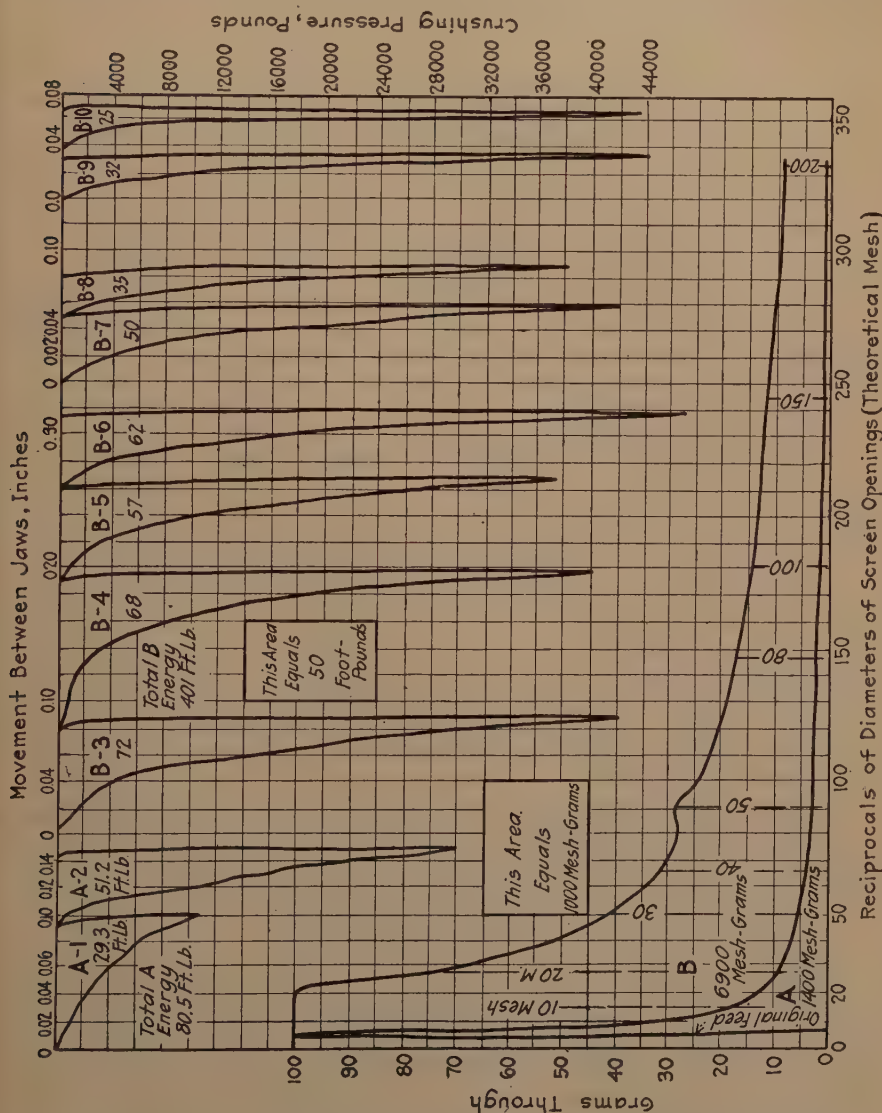


FIG. 12.—CRUSHING-SURFACE AND ENERGY DIAGRAMS. SILICEOUS MEXICAN ORE.

intermediate screening, the purpose being to crack every piece. The energy is represented by the areas A-1 and A-2. It is of interest to note how the release of maximum pressure is usually accompanied by

some movement of the jaw, as in this case, indicating a return of energy to the machine, shown in the diagram by the slight curve back to the zero-pressure line. After the particles adjusted themselves upon release of the pressure it took more movement between the crushing faces to get up to the previous pressure, as indicated by the gap between A-1 and A-2. After sizing and plotting the screen analysis in the form of a crushing-surface diagram A, the particles were re-crushed eight times, each time being screened through a screen of about 30 mesh to remove the fines and eliminate the condition of "choke crushing" as far as possible, the oversize only going back to the machine. The energy areas for these eight crushings are B-3, B-4, B-5, B-6, B-7, B-8, B-9, and B-10; the screen analysis is plotted as a new side of the crushing-surface diagram, B. The areas of the energy diagrams A-1 and A-2 were found by planimeter to be 1.40 and 2.38 sq. in. respectively, a total of 3.78 sq. in. One square inch, for the scales used, being equal to 250 in.-lb. or 20.83 ft.-lb., 3.78 times 20.83 or 80.5 ft.-lb. of energy was expended in making the crushing A. By similar reasoning, the area of the crushing-surface diagram A, as found by planimeter (as far as screening was carried out, to the 200-mesh sieve), multiplied by the value of its unit area in mesh-grams, shows that 1,400 mesh-grams of surface was produced. From

these two quantities, a value of $\frac{1400}{80.5} = 17.3$ mesh-grams per foot-pound (+200-mesh sieve) was obtained, as the constant to be used in calculating efficiencies, etc., for this rock. Note, however, that the mesh-grams entering into this constant are only those "plus" 200-mesh sieve, and that there are perhaps more mesh-grams in the "minus" 200-mesh than in the "plus," and they must at some time be taken care of in calculations. It is these "minus" mesh-grams which will account for the discrepancies in results by Rittinger's theory, and which will not account for discrepancies in results by Kick's law.

18. For B, the total energy expended was 401 ft.-lb., producing 6,900 mesh-grams (+200-mesh sieve) giving a unit value of 17.21 mesh-grams per foot-pound "plus." Fines were screened out after each jaw movement, to reduce "choke crushing."

19. Tests E, F, G, H, J, K, L, M, and N were made upon Bedford limestone, the first in July, 1912, and the rest in July and August, 1913. This is one of the best known building stones in the United States. It occurs in great beds in southern Indiana, is oölitic in texture (weathering brings out myriads of fossils) and, because of its apparent uniformity, is probably as good a rock as could be selected for a standard. (I reserve the right to change this opinion later.) The particular specimens came from stone furnished for the Purdue Library building.

20. The crushing-surface diagrams of E and F (Figs. 5 and 4 in my original article in the *Engineering and Mining Journal* of May 24, 1913) are

not shown here. In E, practically all the material was broken in the machine, whereas in F, a great many pieces between 30 and 80 mesh were not broken, but simply had their edges rubbed off. My records as to F

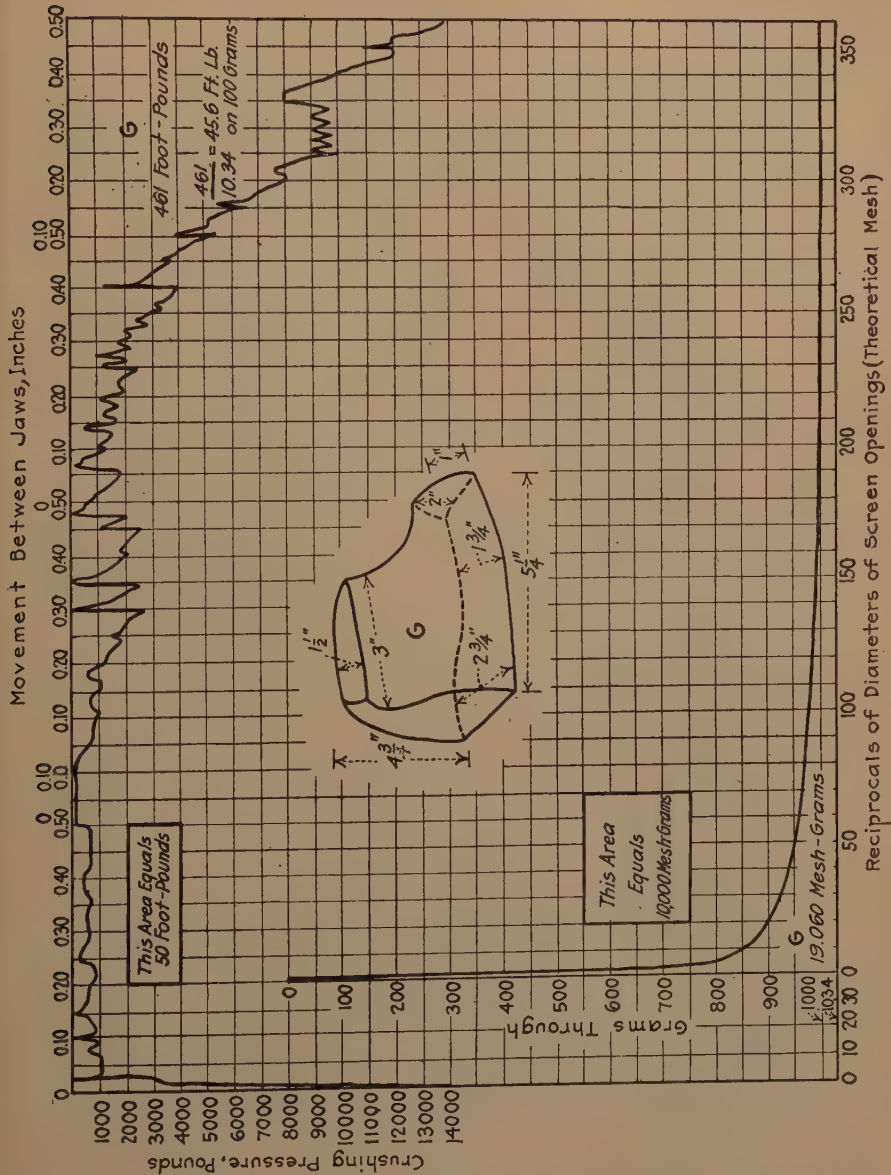


FIG. 13.—CRUSHING-SURFACE AND ENERGY DIAGRAMS, AND FEED-SAMPLE. BEDFORD LIMESTONE.

do not clearly show whether I screened between crushings; this was done in the case of E.

21. Test G was on a block of the dimensions shown in the small

sketch in Fig. 13, having a superficial area of 63 sq. in. and a weight of approximately 1,031 g. The energy curve is rather long drawn out after the first break, and the irregularities in this curve may be explained

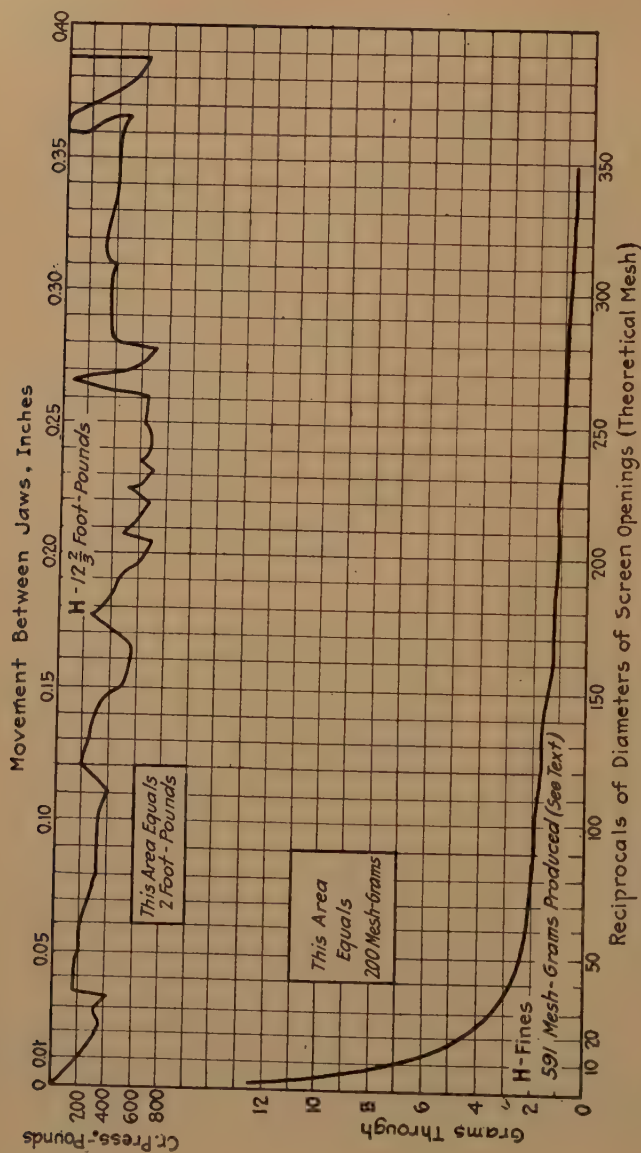


FIG. 14.—CRUSHING-SURFACE AND ENERGY DIAGRAMS. BEDFORD LIMESTONE.

as due to frequent ruptures of fairly large pieces where the load drops back almost to zero. A total movement of nearly 2 in. is recorded; here only one or two pieces perhaps were resisting, while in some of the other

cases a great many pieces were resisting at one time. At the end of the first $\frac{1}{2}$ in. of movement the finer pieces were screened out, and the larger returned to the crushing zone; these pieces being rather flat; about 2 per cent. of -200 mesh being produced.

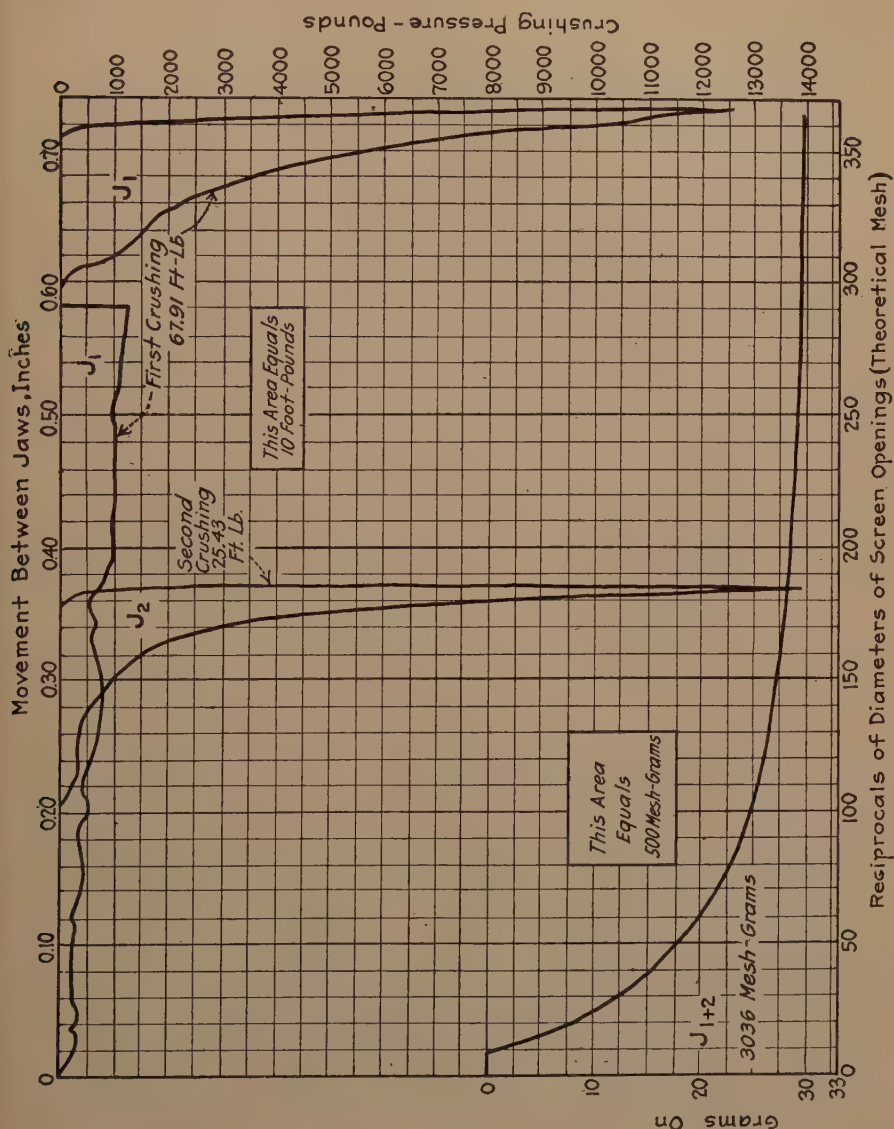


FIG. 15.—CRUSHING-SURFACE AND ENERGY DIAGRAMS. BEDFORD LIMESTONE.

22. H, shown in Fig. 14, covers the results of crushing four pieces having a total superficial area of 30.25 sq. in. and a total weight of 175.5 g. After crushing, eight pieces weighing 163 g. had an area of 30.25 sq. in., just offsetting the original area, while the remaining 12.5 g.

were screened, and the results were plotted as a crushing-surface diagram. 12.66 ft.-lb. were expended in crushing the 175.5 g., equivalent to 7.2

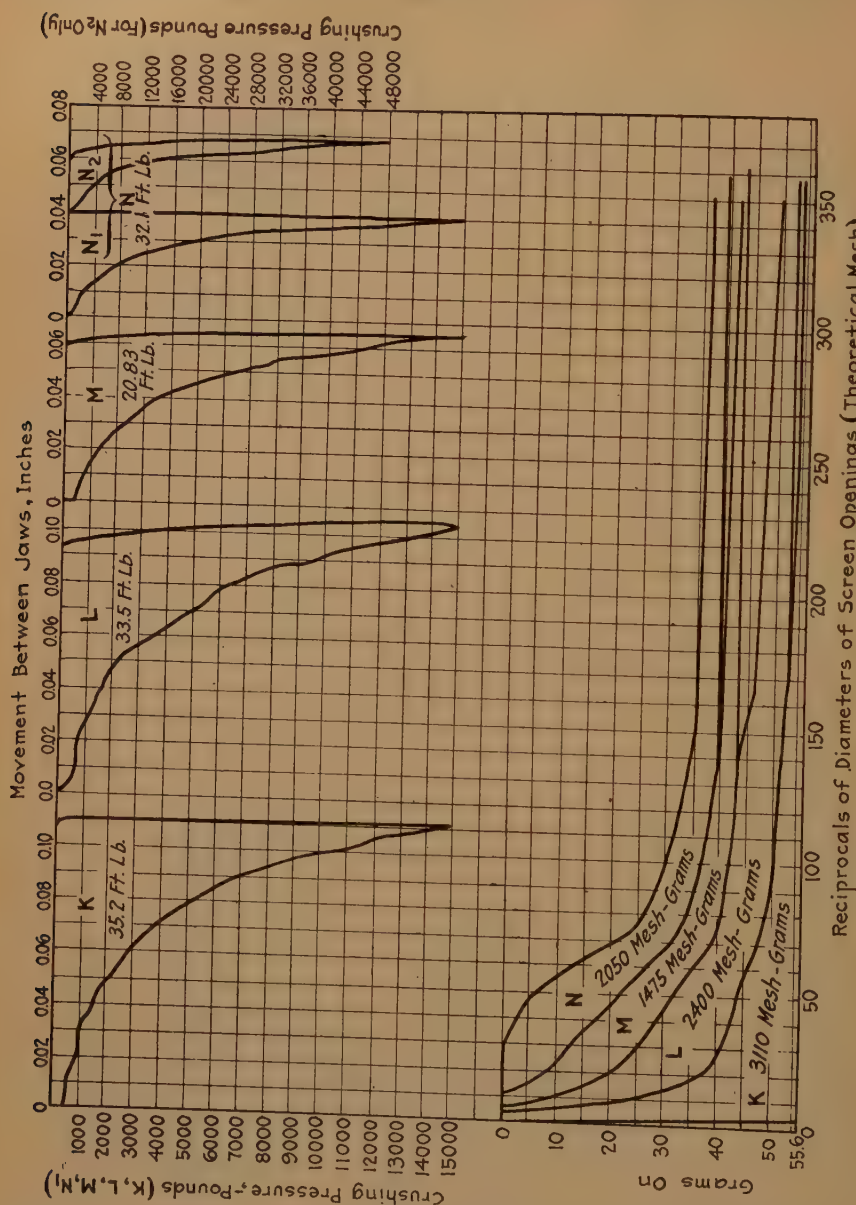


FIG. 16.—CRUSHING-SURFACE AND ENERGY DIAGRAMS. BEDFORD LIMESTONES.

ft.-lb. per 100 g.; the eight pieces noted above were estimated to be the equivalent of 60 per cent. of 1 in., 32.5 of $\frac{1}{2}$ in., and 7.5 of fines, as recorded in the crushing-surface diagram. The energy curve is irregular as

in the case of G and for the same reason, movement was 0.4 in.; there was no intermediate screening.

23. In J (Fig. 15) a single rectangular piece about 1 by 1.25 by 1.25 in., weighing 33 g., was crushed to pass a 10-mesh sieve, with one intermediate screening. After a deformation of about 0.6 in. was recorded

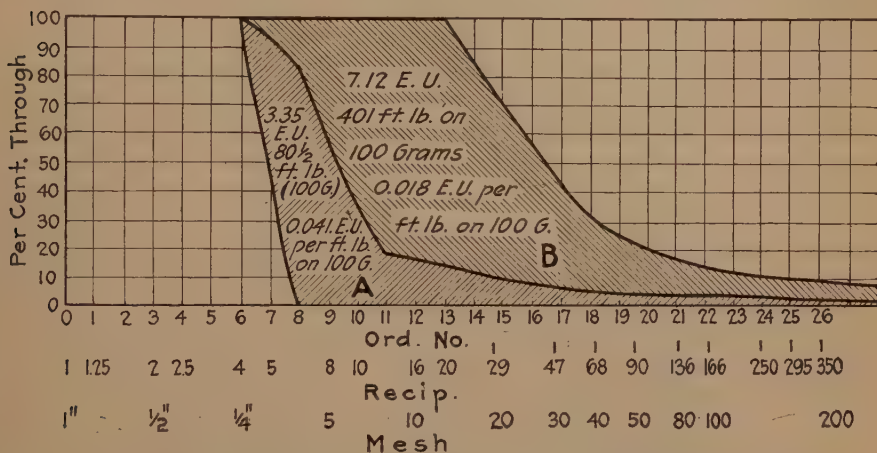


FIG. 17.—MECHANICAL VALUE DIAGRAM. SILICEOUS MEXICAN ORE.

without great pressure, the pressure goes up very rapidly, probably by reason of the packed condition of the particles, a great number resisting crushing. After screening out the -10 mesh, a similar energy curve is obtained on the +10-mesh pieces. The crushing-surface diagram represents the combined product from both crushings. In both cases, the

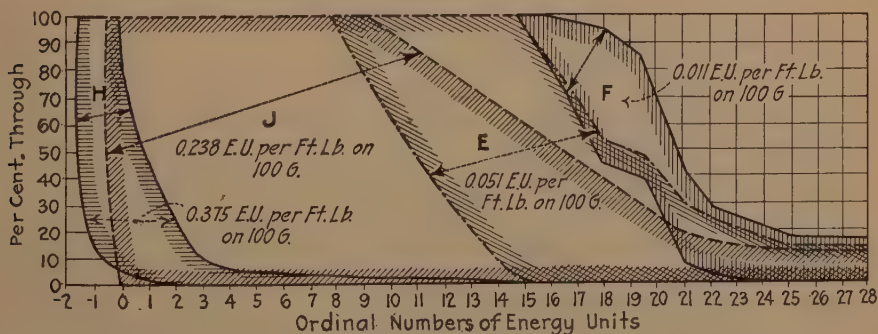


FIG. 18.—MECHANICAL VALUE DIAGRAMS. BEDFORD LIMESTONE.

packed condition, in which the particles rub on one another and produce much -200 mesh, accounts for the low value of the mesh-grams per foot-pound.

24. Fourteen very flat pieces weighing 55.6 g., with about 20 sq. in. of surface, averaging in size about 0.75 in. sq. by $\frac{1}{8}$ in. thick, were sub-

jected to four separate crushing operations, K, L, M, and N, as shown in Fig. 16. The first crushing K was continued until every piece was found to have been broken at least once; screen analysis was then made and recorded in the crushing-surface diagram, and the material +100 mesh was recrushed as L. M is the result of crushing in a similar manner the +80 mesh from L; and N is similarly the result from the +80 mesh of M. Each crushing was carried as far as the record shows without attempting to remove any fines, although in the case of N pressure was released once and then reapplied, as recorded, in an endeavor to apply more energy. The base lines of L, M, and N separate from the discharge or product lines of the preceding operations, showing the smaller quantities of material crushed in each case.

25. In Fig. 17 the results obtained in A and B on the siliceous Mexican ore have been plotted in a mechanical-value diagram. For the coarser crushing A, 0.041 E. U. per foot-pound-applied on 100 g. was the result,

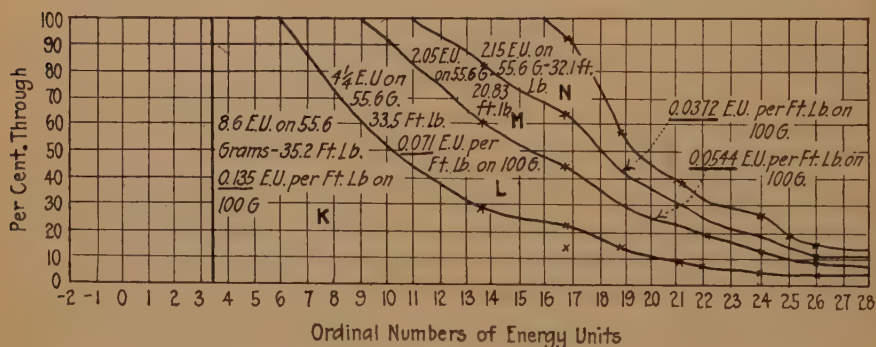


FIG. 19.—MECHANICAL VALUE DIAGRAMS. BEDFORD LIMESTONE.

while in the case of B only 0.018 E. U. (etc.) resulted, 125 per cent. more being produced when the coarser crushing was done. If graduated crushing had not been done, more energy would have been expended upon the -200 mesh; and I believe this discrepancy would then be greater.

26. In Fig. 18, E, F, H, and J (Bedford limestone) have been plotted as mechanical-value diagrams. Note how the E. U. per foot-pound applied on 100 g. decrease consistently from coarse to fine: H, 0.375; J, 0.238; E, 0.051; and F, 0.011. Similarly in Fig. 19, K, L, M, and N are plotted to E. U. diagrams with results decreasing consistently from coarse to fine: K, 0.135; L, 0.071; M, 0.0544; and N, 0.0372 E. U. per foot-pound-applied on 100 g.

27. On another ore, an Arizona porphyry, a series of crushings similar to K, L, M, and N, gave results from coarse to fine as follows: O, 0.055; P, 0.049; Q, 0.0296; R, 0.0149; S, 0.0127; all E. U. per foot-pound-applied on 100 g. The range of sizes was small; the initial feed averaged

TABLE I.—*Summary of Results*

Character of Rock	Mexican Siliceous Ore $\frac{1}{4}$ in.		Bedford Limestone										Porphyry, Mill Feed				
Operation	A	B	E	F	G	H	J	K	L	M	N	O	P	Q	R	S	
Mesh-Grams of Surface Produced (Plus 350 Reciprocal) ..	1,400	6,900	8,800	8,450	19,060	665	3,036	3,110	2,400	1,475	2,050	1,250	1,225	1,470	1,254	1,220	
Foot-Pounds Energy Applied ..	80.5	401	166.3	295.6	461	12.7	93.3	35.2	33.5	20.8	32.1	50	48	56 $\frac{1}{4}$	103.3	75	
Mesh-Grams per Foot-Pound (Plus 350 Reciprocal)	17.3	17.21	53	28.7	41.3	52.5	32.6	33.4	71.9	70.8	63.8	25	25.5	26.1	12.15	16.3	
Stadler's E. U.'s per Foot-Pound Applied on 100 g	0.041	0.018	0.051	0.011	0.213	0.375	0.238	0.135	0.071	0.0544	0.0372	0.055	0.049	0.0296	0.0149	0.0127	
Remarks	Series		Medium		Fine Choke Crushing		Coarse Pieces Choke Crushing		Series		Medium		Coarse		Series		Fine Choke Crushing
	Coarse	Fine			Large Block		Coarse	Coarse							Medium		

about 4 mesh, while all five crushings resulted in the production of not more than 8 per cent. of -200-mesh sieve product. This work was done in December, 1913.

28. Fig. 20 shows the results of plotting an approximate average of feed and discharge for all tests against both the E. U. per foot-pound-applied on 100 g. and the mesh-grams per foot-pound (+350 reciprocal).

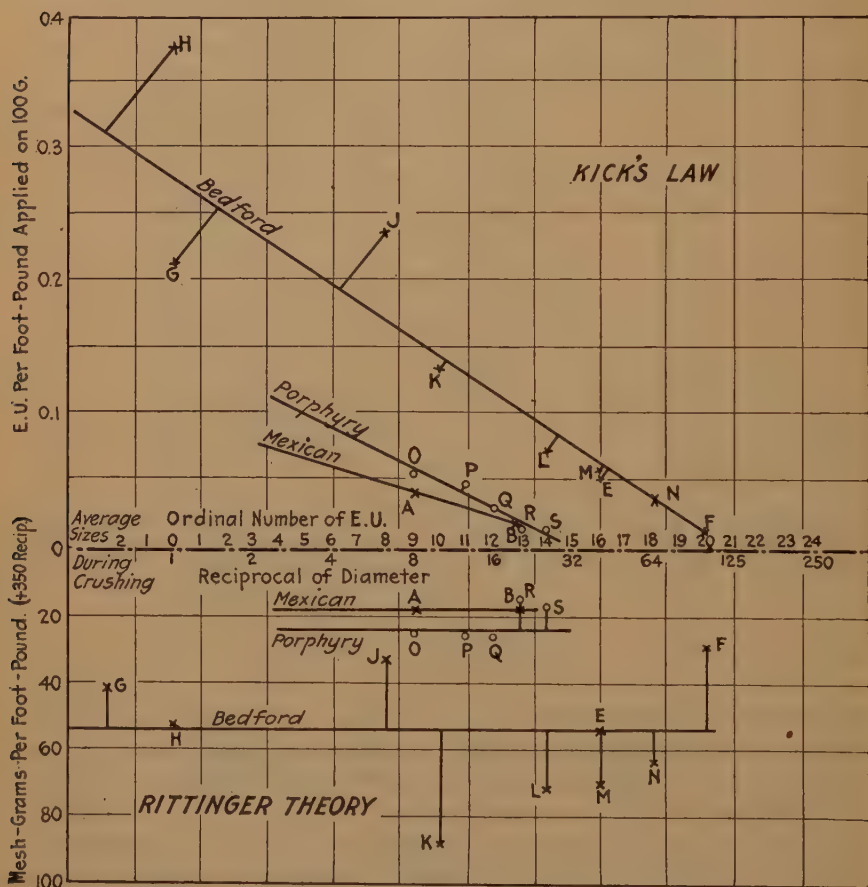


FIG. 20.—VARIATION OF TEST RESULTS BY BOTH METHODS.

The estimated average, taken as near to the center of gravity of the mechanical-value diagrams as possible by ocular observation, gave the horizontal position of each point, while the calculated E. U. and mesh-grams gave the two vertical positions.

In the upper diagram the inclined line (Bedford limestone) connecting the various points shows that for different average sizes of material undergoing crushing, the production of E. U. per foot-pound-applied on 100 g.

decreases from coarse to fine; according to Stadler's interpretation of Kick's law there should be no decrease, they should be the same at all sizes. From the values tabulated in Table I, which is a summary of all these tests and calculated results, it will be found that for the Bedford limestone, the *coarsest* crushing gave 34 times the number of E. U. per foot-pound-applied on 100 g. that the *finest* crushing gave. For the other materials, the graphs incline similarly and in sufficient degree to show the fallacy of Stadler's application of Kick's law.

29. In the lower diagram, the results of this work on a Rittinger basis have been plotted, adhering, however, to the average sizes, taken as described above, which will not affect the results, except to lengthen the curve in the coarse sizes and to shorten it in the fine sizes. Horizontal straight lines averaging the results for each material have been drawn in; if the same number of foot-pounds-applied produce the same number of mesh-grams at all sizes, then the Rittinger theory is correct, and a horizontal line represents it in this diagram. If some of the discrepancies be explained as due to "choke crushing," producing an excess of unmeasurable -200 mesh (-350 reciprocal), the majority of points will come nearer to a horizontal line, and better confirm our belief in the Rittinger theory. The results clearly show that the Rittinger theory is so much more nearly correct for crushing operations, that I believe we can accept it and use it for the determination of crushing constants, for machine comparisons, for efficiency tests, etc.

30. Since it is obvious that the sizes finer than our ordinary screens will separate are important if the Rittinger theory be correct, finer screens or classification followed by microscopic measurements will probably be necessary for accurate determinations. To some extent at least, we can approximate the quantity of the finer sizes; there seems to be a law connecting the quantities of different sizes, that the discharge line of a crushing-surface diagram from a uniform crushing operation will be in the form of a hyperbola, and, when plotted on logarithmic paper, as in Fig. 21, will appear as a straight line. In Fig. 21 the product line of N indicates that it would have some 4 per cent. of 1,000-reciprocal size and some 0.6 per cent. of 10,000-reciprocal size; they have surfaces respectively equivalent to 40 per cent. of 100 reciprocal, and 60 per cent. of 100 reciprocal, taken separately; and if these values are correct, these sizes are fully as important in the consideration of energy disposal as all of the +100-reciprocal material. The law of the finer sizes might be something of this form:

$$RW^x = C$$

R being the reciprocal of the diameter (theoretical mesh) under consideration; W, the weight of the material passing through a screen of this diameter opening; and x, a constant depending upon the character of the

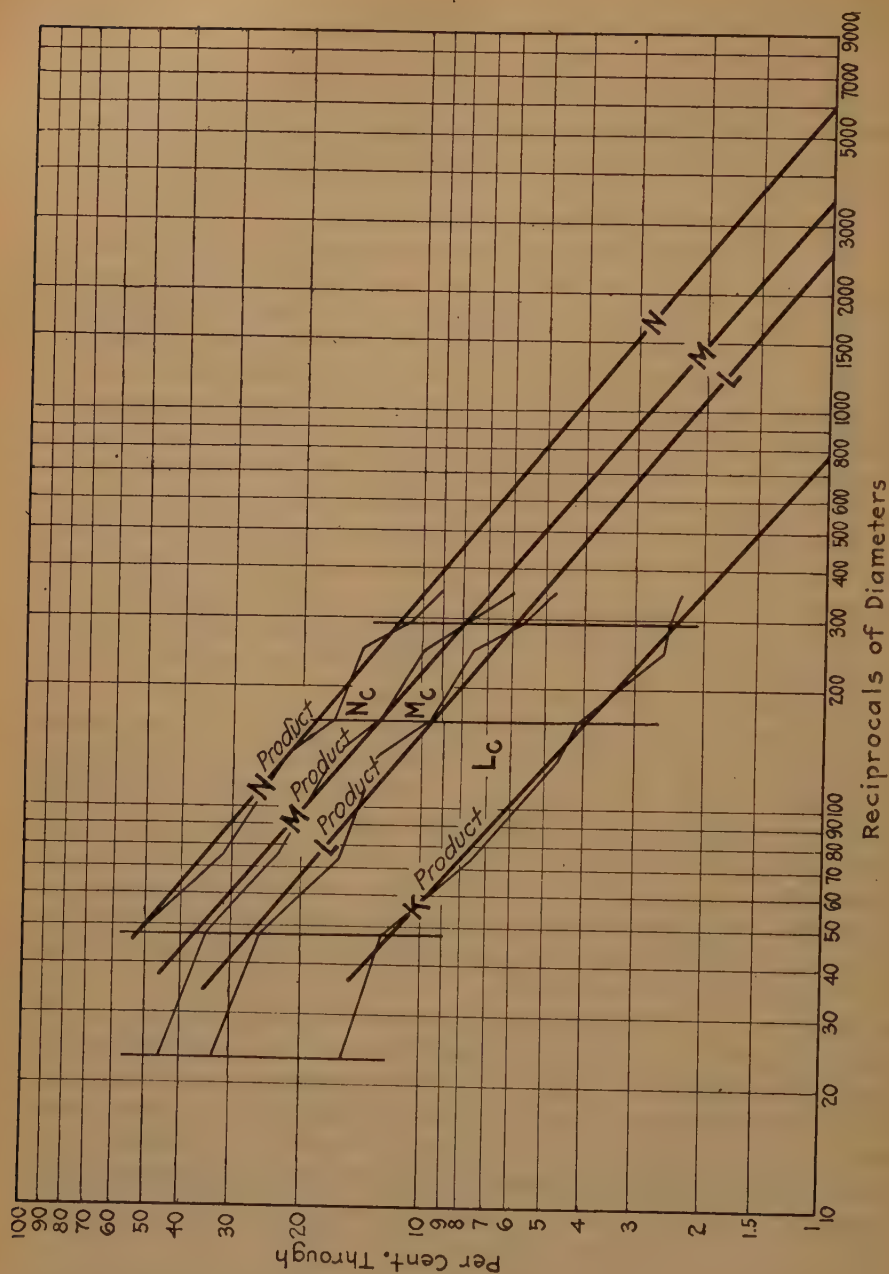


FIG. 21.—SCREEN ANALYSES PLOTTED TO LOGARITHMS.

ore, the type of machine in which crushing is done, and possibly upon the screen analysis of the feed; and C, the number of crushing units (mesh-grams, mesh-tons, etc.) in the undersize, considered as all of that one size, and proportional to the area between the co-ordinates of the point on the curve (intersection of C and W) and the zero lines.

This, however, is largely speculative, and is only suggested because it may be of some value to some future investigator.

SUMMARY OF RESULTS

31. Table I gives a summary of results. It will be noted that in most cases the high values for mesh-grams per foot-pound were obtained with low values in foot-pounds, and *vice versa*. This seems to indicate that the highest values and therefore the most efficient crushing would be obtained when each particle was just cracked; a thing that is generally understood to be true in connection with rolls, for example, free crushing being, within limitations, more efficient than choke crushing. Choke crushing here gave uniformly low results, as might be expected. In the case of B, the several crushing operations involved small amounts of power, but they were lumped together for final screen analysis. If all the power had been applied at one time, without intermediate screening, there would not have been such a nice agreement in the mesh-grams per foot-pound, between A and B.

32. As the writer sees the situation, after the acceptance of the substantially correct theory of crushing, there must be recognition of the fact that rocks of varying characteristics are met in crushing operations, that these rocks offer different resistances to crushing, and that it is unfair to compare the operations of two crushing machines upon dissimilar ores (even if they look alike) until some values are given to the resistance or energy absorption of these rocks under crushing conditions. Probably these units will be determined practically by applying a definite amount of energy to a definite weight of the rock, all of a predetermined standard size. The work of determining these constants should be done thoroughly by competent hands. The matter of checking my conclusions can be accomplished perhaps in several of the school testing laboratories of the country. No elaborate apparatus, and apparently no very high degree of accuracy, will be required to prove that Kick's law is of no use to mill operators and engineers.

33. This paper has been prepared under some disadvantages which may account for the possible lack of logical arrangement. The matter was originally submitted to the committee of the Institute in May, 1914, and upon their suggestion the paper was enlarged, corrected, and rearranged. The writer acknowledges his indebtedness to their suggestions. Acknowledgment should also be made of the courtesies extended by

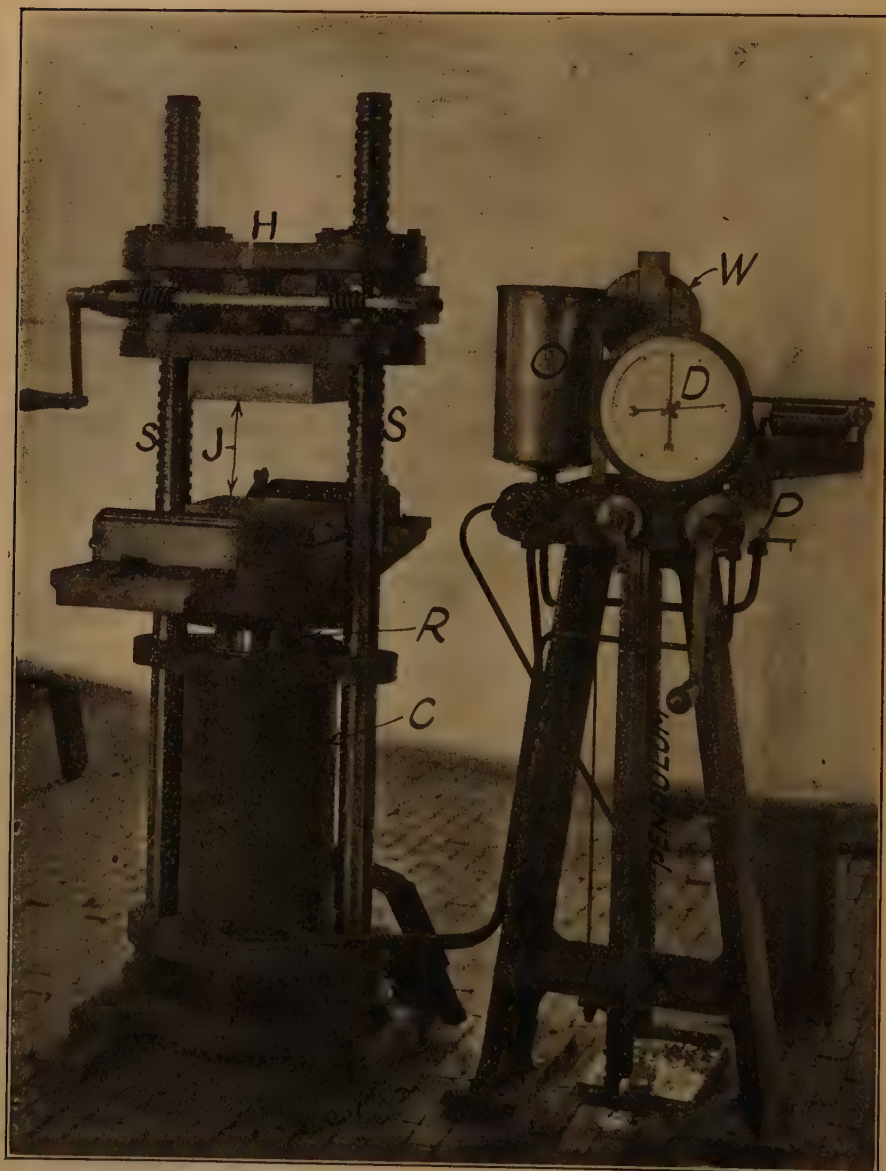


FIG. 22.—AMSLER-LAFFON TESTING MACHINE.

Dr. W. K. Hatt, head of the department of Civil Engineering, and Professor Schofield, at Purdue University, who allowed the writer free run of the laboratories for Testing Materials.

APPENDIX

The Amsler-Laffon hydraulic testing machine, with which these experiments were made, is shown in Fig. 22.

The material is crushed between the faces of the jaws *J* by pressure applied against the ram *R*, contained in the cylinder *C*. The head *H* carrying the upper jaw can be moved up and down the fixed screws *S* by nuts having worm-wheel threads cut on their outer surface into which mesh two worms driven by the hand crank shown to the left.

The pump *P* for supplying the pressure to the ram cylinder *C* is operated by a hand screw on the side of the frame carrying the registering and auxiliary mechanism. Suitable piping connects the pump with the cylinder, the supply tank *O* and the registering cylinder *X*. From *X* suitable links and levers connect to the pendulum and adjustable weight *W*, the swing of the pendulum, which is proportional to pressure exerted, being recorded on the drum or indicated upon a dial *D*. In all of this work the readings of the dial were used. With different settings of the weight *W*, four different scales of pressures having maximum values of 15,000, 50,000, 100,000 and 150,000 lb. may be used, suitable dials being provided.

The deflectometer, which is a part of the regular laboratory equipment, consists of a lever carried at its fulcrum in a suitable base. At the short end of the lever a fine-thread vertical adjusting screw can be brought into contact with the upper jaw *J* of the machine, while the long end of the lever fits against a vertical scale, and moves a vernier with it as it moves up and down, thus permitting the measurement to 0.001 in. of the movement between jaws.

DISCUSSION

ALGERNON DEL MAR, Los Angeles, Cal. (communication to the Secretary).—Since reading the first discussion of the work done in crushing, I was impressed with the possibilities inherent in any law that would give us the means of comparing different machines crushing the same ore. It is manifestly impossible to compare two machines crushing different ores unless the specific resistances of the ores are known, and it is likewise inadequate to compare two machines crushing the same ore through radically different meshes, for the resistance to fracture varies in the same ore with the degree of fineness, so for this particular phase of comparison we must experiment on ores crushed through nearly the same range of sizes.

After a diligent study of the literature on the subject I concluded that while no exact law could be formulated to represent the work done in crushing, a close approximation was possible on the assumption that the work done in crushing was proportional to the surface exposed by the crushing force and not to the volume at every stage, and thus I came to uphold Rittinger. Having at my disposal no means to test the accuracy of the two theories by actual experiment, I could only reason from a mathematical standpoint. Stadler and Taggart, both reasoning from the same standpoint, proved to their satisfaction the falsity of the Rittinger theory and the correctness of Kick's law which holds that the work done in crushing is proportional to the volume at every stage. We waited for a practical demonstration to prove the worth of either theory and Professor Bell, at McGill University, sent word that by actual demonstration Kick's law was proved to be correct; to my knowledge the figures were not made public, and now the author of the paper under discussion has proved that Rittinger was correct after all. Taggart laid emphasis on the point that in the Rittinger theory the distance factor in the general formula—work = resistance \times distance—was not taken into account but it will be noticed that Mr. Gates in each experiment has recorded the distance through which the crushing force acts.

In 1914 I attended a meeting of the Institute in New York at which Professor Taggart's paper was presented but not read; there was no discussion. Last spring I read a paper on the same subject before the Los Angeles Section of the Institute, but the discussion drifted off into the degree of fineness to which a rock breaker could be made to grind economically, and I pointed out to the members present that by the use of the Rittinger formula it was possible to determine this limit. I mention these facts here to indicate that unless we are able to show the practical import of a law for crushing, it is impossible to arouse interest in the subject.

It appears strange that while Stadler regards the Rittinger theory as pure nonsense, and Taggart argues that the distance factor is not considered, Gates proves the theory by experiment. Likewise, there is the anomaly that mathematically both laws can be proved correct according to individual ideas of how the crushing forces act.

Although I have lacked the advantage of a physical laboratory in which to make a practical demonstration, I have in a crude way made some experiments which appear to substantiate my original postulate that the work done in crushing is proportional to the reciprocal of the diameters to which the ore is crushed. Stadler in 1910–11 brought forward his energy units (E. U.) as a means of computation; in 1912 I presented the method of computing crushing efficiencies from the reciprocals of the diameters, and in 1913 Gates used his crushing-surface diagrams to demonstrate the same computations. The crude experiments I refer to were made with a

small stamp on concrete blocks with an artificial plane of weakness through the center of one face; by which means I demonstrated to my satisfaction that the work done in crushing is proportional to the surface exposed. These experiments are recorded in the paper read before the Los Angeles Section and need not be repeated here. For those who have not a clear conception of the two formulas involved in this discussion it might be well to cite them for comparison.

The Rittinger formula for the mechanical units used in crushing is D/d , and while Stadler gives the formula for his energy units as $-\log S$, where S is the diameter of the screen aperture, a simpler formula is $\log D/d$. The Rittinger formula is supposed to represent the work done at the moment of fracture.

As stated by Gates, the Kick formula represents the work done within the elastic limit. We then have some phenomenon at the elastic limit which will account for the difference. The point for future study is here indicated, and the question whether this difference can be accounted for in the work returned by the crushing machine on the release of the crushing force, the heat developed by compression and dissipated at the moment of fracture, or by other physical changes, affords opportunities for original research in this field.

The work done in crushing is generally considered to be an academic subject with no practical application, and only by demonstrating that this is not true can interest in the subject be awakened. At the Winona mill in the Lake Superior district two Hardinge mills, one 6 ft. in diameter and 5 ft. straight face, the other 8 ft. diameter and 30 in. straight face, are run side by side on the same ore of the same degree of fineness. It is desired to know, for future installations, which mill is the more effective. H. H. Seeber, the manager, prepared a chart showing the power consumed by each mill, screen analyses of the feed and discharge, and the tonnage, and from these data it was found that the smaller diameter mill was doing better work. This shows one practical application of the laws of crushing. Had this demonstration been carried far enough it might have been found, by the same laws, that the size of the feed for the two mills favored the one of less diameter and that by feeding a coarser product the bigger diameter mill would be proved most efficient. This shows another application of the law, *i.e.*, a means of determining the best size of feed or discharge for a crushing machine.

In this connection may be mentioned a recent episode which, though a little removed from the subject under discussion, may be of interest. Stadler has calculated the relative efficiency of the tube mill and stamps on the Rand to be 17 to 70 (omitting fractions). In passing to and fro before the discharge end of a tube mill one is often impressed with the great increase of heat above the atmospheric temperature, and it appeared to me that in some way this conversion of mechanical work into heat

might give us another method of comparing efficiencies. For the purpose of obtaining some figures I crudely measured the difference in temperature at the feed and discharge ends of a tube mill working in the district and found it to be 4°C . To determine more fully the bearing of this formation of heat I called upon another mine superintendent and stated my desire to measure the difference in temperature at the two ends of his mill. He gave me permission, but impressed upon me the fact that the subject was purely academic and had no practical import. I found the difference in temperature at the two ends of his mill to be 10°C . and, as might be expected, the differences in temperatures of the two mills bore a close relation to their output. I brought this to the manager's attention, thus clinching my contention that while the subject is no doubt academic, by following its ramifications some interesting and useful hints that have practical application might be found. If all the heat generated in a crushing machine could be measured, knowing the specific heat of the substances heated we might be able to settle Kick vs Rittinger.

We must thank Mr. Gates for his untiring energy in endeavoring to fathom this subject, and while I am convinced of the soundness of his arguments, we should like to see Professor Bell's figures and likewise hear from our own State University on the subject.

H. STADLER, London, S. W., England (communication to the Secretary*).—The laws that the change of stress and change of configuration are proportional, and that the work spent upon producing these stresses is proportional to the *product* of change of stress into change of configuration, are accepted by physicists and technologists all over the world. Expressed as above they are general and refer to all forms in which energy appears (dynamic, kinetic, electric, heat, chemical affinity, etc.) and to all sorts of changes of configuration (elastic deformation, plastic deformation with rupture as the ultimate end stage, displacement, change of state, etc.). For the sake of convenience these laws are expressed in a different way in each of the various branches of technology, using the terms commonly employed in that particular line. In statics and mechanics, for instance, it is said that the force, or the breaking load, varies as the square of the homologous dimensions; and that the mechanical work is proportional to the *product* of force into the distance through which the force has to move against the opposing stresses. These laws are well understood and are familiar to all engineers. Applied to crushing we may accept Kick's version which correlates the energy required for causing the rupture of a body with the reduction of its volume (or weight). According to the so-called Rittinger theory, which is offered in opposition to the above laws, "work" and "force" vary at the same

* Received Oct. 16, 1915.

ratio; advocates of this hypothesis disregard the distance factor. Its real position becomes apparent whenever it is subjected to practical tests.

All branches of technology abound with examples of the universal validity of Kick's law, or of applications of it; we may take it either way. The law holds good in extreme cases, from the coarsest possible crushing by blasting down to the elastic deformation of gases and liquids. The so-called Rittinger's theory, on the other hand, breaks down by simply glancing at a grading analysis, for instance, of the battery pulp given below:

Battery Pulp, 9 Mesh (0.272 in.)

Grading		Kick's Law			Rittinger's Law		
I. M. M.	Weights, Per Cent.	Volume Factor	Efficiency of Stamp		Surface Factor	Efficiency of Stamp	
			A	B		A	B
+ 5	5.0	9	0.450	Reduced to 100 per cent. 15.214	10	0.500	Reduced to 100 per cent. 82.068
+ 8	14.8	11	1.628		16	2.368	
+ 12	12.4	13	1.612		24	2.976	
+ 20	11.1	15	1.665		40	4.440	
+ 30	9.0	17	1.530		60	5.400	
+ 50	9.7	19	1.843	0.0	100	9.700	0.0
+ 80	7.5	21	1.575		161	12.075	
+ 120	4.5	23	1.035		238	10.710	
+ 200	4.9	25	1.225		400	19.600	
- 200	21.1	28	5.908		645	136.095	
100.0			18.471	15.214		203.864	82.068
Ratio:			100	82.37		100	40.25

Assuming for argument's sake that by any means (yet to be discovered) the production of the 21.1 per cent. of the -200 grade could be prevented in a second stamp (with a proportional increase of the percentages on the remaining grades), the volume factor taken as a standard for the measurement of the energy consumed in the two cases would account for a lower power consumption of 17.6 per cent. for the improved stamp, and the surface factor of 59.8 per cent., which latter figure is obviously absurd.

In his original article Mr. Gates advances a "theoretical proof of the Rittinger theory," claiming that if a cube be ruptured by a shearing force applied by two offset faces, this force would act directly in the vertical fracture plane, "the mass of the cube away from this surface receiving practically no pressure or deformation." The fallacy of this reasoning has been exposed by several writers in leading technical journals, but Mr. Gates reprints his "theoretical proof" *in extenso*.

Another argument much abused by the opponents of Kick's law is the contention that it refers only to elastic, but not to plastic, deformations. These two critical points are, in highly elastic materials such as are generally subject to crushing, so close together that this objection could be discarded as having no appreciable effect on the results, but Kick's law does not need to have recourse to such an allowance, for a simple household sausage machine is quite good enough for demonstrating experimentally that the work spent upon causing the analogous plastic deformations is proportional to the volume, or weight, of the meat paste.

Mr. Gates opens his argument by pointing out that the ratio of the energy expended for breaking a 16-in. cube in two stages, first to 1-in. cubes and then to $\frac{1}{16}$ -in. cubes, is 1:1 for Kick's law and 1:16 for Rittinger's law. With such a clear issue one would have expected that an experimenter would have proceeded, with the excellent apparatus at his disposal, to measure the work required for crushing varying sizes of geometrically and technologically similar bodies of any shape. Instead of carrying out such simple tests, Mr. Gates engaged in experiments made with mixed sizes, an undertaking beset with difficulties which he increased by introducing all sorts of easily avoidable complications.

One of these is the creation of a new unit, the "mesh-gram," which in a rather cumbersome definition is said to mean the area of surface exposed in reducing 1 gram of particles of equal size to a smaller size, this area to be measured by the difference of the reciprocals of the diameters. The reference to weights is utterly superfluous in a case where we have to deal only with relative values for which the reciprocal of the diameter is the natural representative.

Other complications which make a scrutiny of the data presented practically impossible are: The use of different materials, the confused and prolix way of expressing simple facts and relations, the tampering with samples by the occasional and unsystematical screening out of the "fines," the too small scale of the tests in dealing with mixed-sized products of which a sufficient amount should be available for arriving at fair average values, and, last but not least, the complete neglect to take the precautions necessary to secure the strict fulfillment of the requirement of analogous conditions in all stages of operation.

Under these circumstances it would serve no useful purpose to go into the details of the tests, and I content myself by showing how easily a blunder can be made if the experimenter, undertaking to overthrow a universally recognized law of nature, is under the influence of the prejudice that "no elaborate apparatus and apparently no very high degree of accuracy will be required to prove its fallacy."

No objection can be raised against an attempt to check the merits of the crushing laws by tests approaching practical working conditions, provided that the conditions be in all crushing stages proportionate and in

accordance with the premises common to all crushing laws. That this was not the case in the present tests is evidenced by the enormously more rapid increase of pressure required for causing an appreciable compression of the crushing beds, when the finer sizes came to be dealt with. In fact, if no "fines" had been screened out, a point would soon have been reached where even the greatest pressure would have had no more crushing effect. Mr. Gates correctly inferred from this observation that free crushing is more efficient than choke-crushing, but he failed to realize that in his tests choke-crushing took place to a much larger extent in the fine- than in the coarse-crushing stages. The temporary remedy of screening out "fines" at intervals did not prevent choking taking place in the immediately preceding test by the fragments produced during the operation of crushing. The conditions of the tests were obviously not analogous in the series of experiments and this operated to the detriment of the efficiency of the fine-crushing stages. The results are therefore not such as warrant drawing inferences on the correctness of a crushing law which conversely is expected to be the theoretically correct standard for the measurement of the efficiency of crushing appliances under varying working conditions.

The great advance of technological knowledge attained as the result of evolution during centuries of honest and thorough research work, to which great (mostly French) scientists devoted their lifetime, is recorded in an abundant literature, and it is not too much to expect that mining engineers who propose to lecture or write on these matters should make themselves acquainted with it and abandon the practice of drawing their wisdom from antiquated text-books on ore dressing.

The Concentrator of the Timber Butte Milling Co., Butte, Mont.

BY THEODORE SIMONS,* E. M., C. E., BUTTE, MONT.

(San Francisco Meeting, September, 1915)

I. INTRODUCTION

PERMISSION to present this paper at the February, 1915, meeting of the Montana Section of the American Institute of Mining Engineers was liberally granted by W. A. Clark, Jr., President and General Manager of the Timber Butte Milling Co., and by the representative of the Minerals Separation Co.

For the many courtesies shown the writer on his visits to the mill and for information supplied, he is indebted to all the officers of these companies, especially to W. N. Rossberg, W. D. Mangam, Hamilton Cooke, Jr., T. M. Owen, T. C. Wilson, R. McGillivray, L. L. Quigley, C. Bartzen and others. The writer takes this opportunity to herewith express to them his appreciation and sincere thanks.

Location of Mine and Mill

The concentrator of the Timber Butte Milling Co. was built to treat the ore mined at the Elm Orlu mine of the Elm Orlu Mining Co., which carries zinc and lead in sphalerite and galena; copper in bornite, chalcocite, tennantite, and tetrahedrite; and gold and silver either free or chemically combined with the sulphides of the base metals. The gangue is chiefly quartz, barite, fluorite, rhodonite, and rhodocrosite. The iron in the ore occurs as iron pyrites.

The Elm Orlu mine is situated in Butte, Silver Bow County, Mont., near the northern border of what is locally known as the Butte sulphide-copper belt, about one mile north of the Anaconda mine, and adjoining the Black Rock zinc mine of the Butte & Superior Co. on the southwest. The mill is on the northern slope of Timber Butte, a prominent topographical landmark in the Butte district. This site permits to a large extent the gravity system of ore flow; it presents a convenient and extensive area for a tailings pond; and it is comparatively near to the source of water supply and to existing railroads.

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The mill is 4.5 miles from the mine in an air line, or 13 miles by rail. The ore is loaded on railroad cars at the mine, taken over the B., A. & P. Ry. high line to Rocker, and thence by the Chicago, Milwaukee & St. Paul Ry. to the Timber Butte exchange, a siding on the Milwaukee

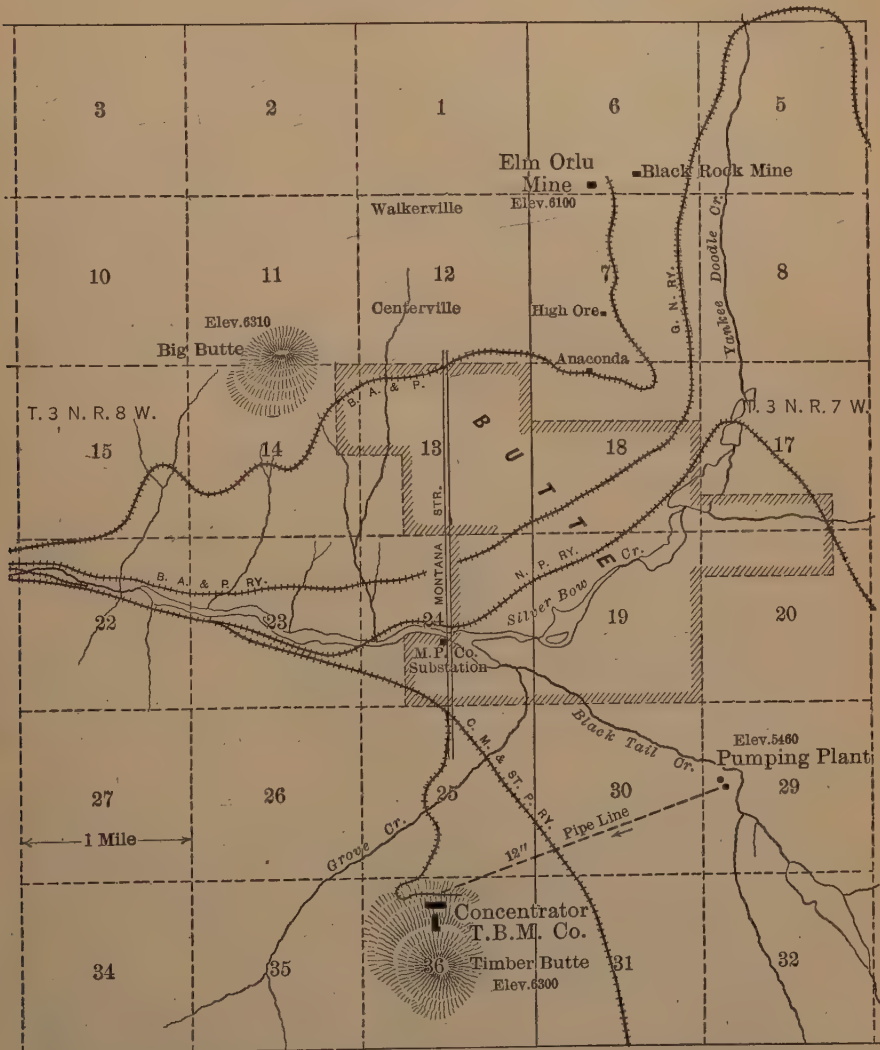


FIG. 1.—MAP OF BUTTE AND SURROUNDING TERRITORY, SHOWING PLANT OF TIMBER BUTTE MILLING CO., AND ORE AND WATER SUPPLY.

line where it crosses Montana Street, a distance of 11 miles. From this siding the cars are hauled, two at a time, to the mill by an electric 400-hp. Baldwin locomotive, a distance of 2 miles. The concentrates are taken from the mill to the sidings on the Milwaukee or the N. Pacific

line where the respective railroad companies receive them for transportation to the smelters.

Fig. 1 shows the location of mine, mill, railroads, and pumping plant.

Design of the Concentrator

The treatment of the Elm Orlu ores presented the usual problem of making a series of products the combined market value of which, after deducting mining and milling costs, would leave the maximum of profit.

The solution of this problem calls for two essential features in a mill: simplicity in the processes employed and economy in operation. Both these features are conspicuous in the plant of the Timber Butte Milling Co., the design of which was based on the results of extensive tests and experiments, covering a period of seven months. In the course of these tests, carried on by a trained staff under the direction of W. N. Rossberg, the company's chief metallurgist, various processes and machines were tried out, the ultimate aim being to make at the least expense the following marketable products:

1. A product for the copper smelter, containing, as nearly as possible all the copper, with some iron and a minimum amount of lead, zinc, and insolubles.
2. A product for the lead smelter, containing, as nearly as possible, all the lead, with some iron and a minimum amount of copper, zinc, and insolubles.
3. A product for the zinc smelter, containing, as nearly as possible, all the zinc, with a minimum amount of copper, lead, iron, and insolubles.

The gold and silver contents of the original ore were permitted to go with each of the above products in varying amounts, as shown in Table I.

During these tests, treatment costs and percentages of extraction were carefully estimated and balanced against each other. The result is the present design and flow sheet of the plant, which by a judicious process of elimination was reduced to two essential operations: table concentration, followed by flotation, the latter in accordance with the Minerals Separation Co.'s patents.

In order to arrive at the most suitable or profitable flow sheet for the Timber Butte mill, the peculiar composition of the Elm Orlu ore had to be taken into consideration. In the first place, there is not sufficient lead in this ore (0.75 per cent.) to warrant a large expense for its recovery. It is, moreover, so intimately combined with the zinc and other sulphide minerals, down to the finer sizes, that it is difficult to make a separation. The percentage of the iron in the ore (3.29 per cent.) is likewise hardly high enough in proportion to the zinc (18.52 per cent.) to permit a satisfactory separation by gravity concentration; yet it is sufficiently high to

reduce the grade of zinc concentrate very materially, if not removed. A large proportion of the iron is combined with the zinc and copper, which reduces the possibility of separating the iron from the other metals, and limits the expenditure which may be allowed for its removal. It therefore becomes necessary to provide at moderate cost a lead-iron product rich enough for re-treatment. This is accomplished on Wilfley and James roughing tables.

These tables also make a finished zinc concentrate which has averaged 3.73 per cent. of insolubles for the six months up to Dec. 1, 1914. This combined with the fact that by flotation and final separation the insolubles in the concentrates are reduced to 5, 4, or even 3 per cent. eliminates any possibility for jigging operations, at least for coarse sizes, since it is difficult to pick out from the ore any zinc particles of, say, $\frac{1}{2}$ -in. size, containing less than 10 or 12 per cent. of insolubles. This is due to the presence of small bands and crystals of gangue material, not visible to the naked eye. Even at $2\frac{1}{2}$ -mm. size, few particles containing zinc, contain less than 8 or 10 per cent. of insolubles. Jigging is also objectionable on account of the resultant high percentage of insolubles in the zinc concentrate, the large consumption of water, and the large floor space required by jigs having a given capacity; all of which features add heavily to first cost of plant and to operating costs.

The recovery of 25 or 30 per cent. of the total original zinc in the ore on the roughing tables, is satisfactory from any point of view. The cost is small, the water consumption low, and the capacity large. The economy of this operation becomes still more apparent when it is considered that on these tables the lead and iron are enriched to a point where re-treatment and the separation of the lead and iron from the zinc become economically possible.

Were it not for the fact that by the skill of the company's engineers the efficiency and economy of the roughing operations were brought to a very high state of perfection, it would unquestionably have been more profitable to eliminate all gravity concentration in favor of a simple flotation plant. For it was found that with the exception of the roughing operations, no other methods of gravity concentration could compete with the flotation process in the cost of producing a certain grade of zinc concentrates.

No commercial method, other than smelting, has yet been found for an efficient separation of the copper from the zinc. The company's engineers, however, are making experiments with a view to accomplishing a satisfactory separation of both the copper and silver from the zinc.

The mill proper consists of three separate buildings, connected by runways for conveyors and concentrate launders. The lower building, 155 by 70 ft. in size, contains the concentrate plant and bins, the coarse-ore bins and coarse breakers, the boiler plant and coal bins. The next

higher building, 260 by 80 ft., contains the roll-crushing, table-concentration, and flotation plants. The highest building, 20 by 30 ft., contains the supply bin and feeders. All three buildings are frame structures on concrete foundations with concrete floors. The main office (36 by 45 ft.); the assay office (42 by 52 ft.) a 127 by 30 ft. build-

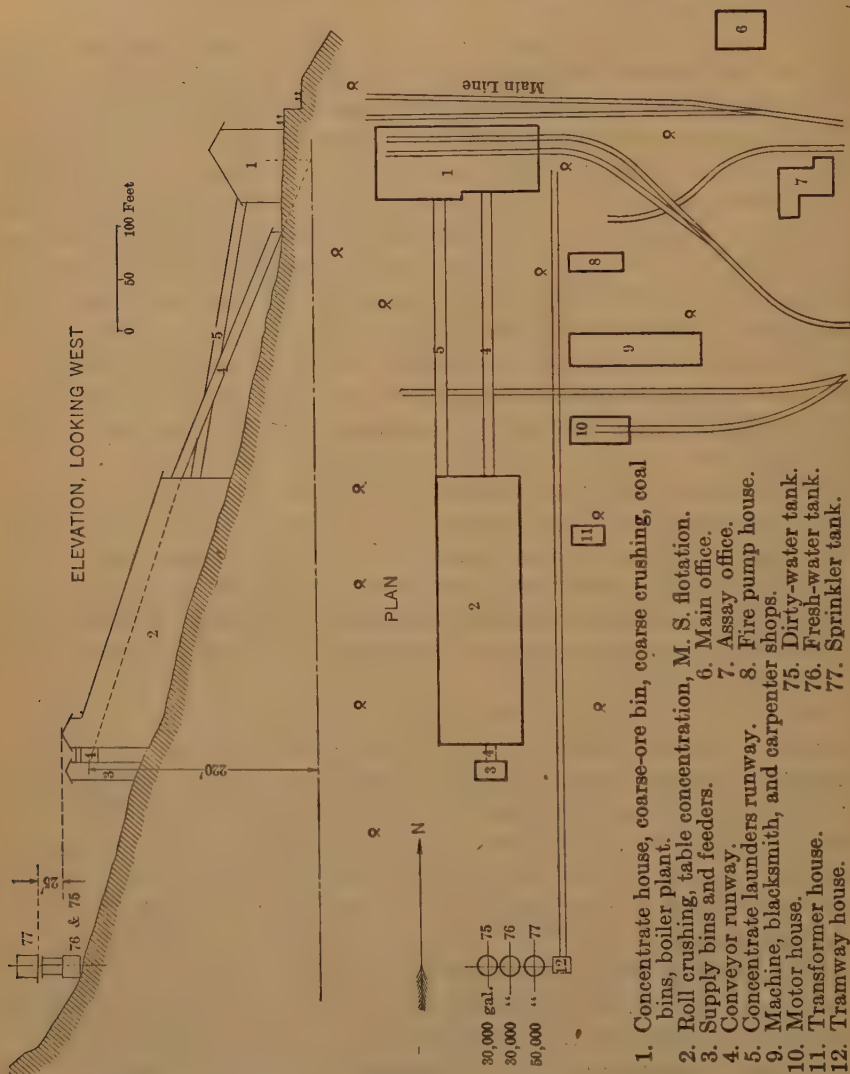


FIG. 2.—PLANT OF TIMBER BUTTE MILLING CO.

ing, containing machine, blacksmith and carpenter shop, and the 57 by 25 ft. motor house are frame structures on concrete foundation. The 18 by 50 ft. building for the fire pump and the 20 by 39 ft. transformer building are of concrete throughout.

An inclined surface tramway, 800 ft. long, runs from the bottom to the top of the mill, along the east side of the buildings, delivering supplies on any level. It is operated by a 37-hp. electric hoist and is good for a 10-ton load.

Fig. 2 shows the location of buildings, tanks, fire hydrants, tramway and railroad tracks.

The plant will be described under the following titles: Flow Sheet; Operation Details; Power; Water Supply; and Fire Protection.

II. FLOW SHEET

(See Figs. 3 and 3A)

1. *Coarse Crushing*

Upon arrival at the top of the lower mill building, the ore is dumped from the railroad cars into two crude-ore bins (1) having a combined capacity of 750 tons. From the bins it goes to a 15 by 30 in. Farrel jaw breaker (3), by way of a shaking-screen feeder (2) with openings 1.5 in. square, and an electro-magnet which removes broken steel, nails, etc. After being crushed to 3-in. maximum size it joins the undersize of screen (2) on its way to elevator (6), which delivers it to a 3 by 8 ft. Symons pulsating screen (7) with $\frac{3}{4}$ -in. round openings. Oversize passes under an electro-magnet, then through a Symons 36-in., style C disk crusher (8), where it is reduced to $\frac{3}{4}$ -in. maximum size. It then joins the undersize of screen (7) on its way to belt conveyor (9).

The conveyor, 630 ft. long, is in two sections, each driven by a separate 20-hp. motor. It travels at a speed of 196 ft. per minute. The vertical lift from bottom to top is 220 ft.

2. *Supply Bins and Feeders*

On reaching the top of the mill, the ore is dumped into a secondary supply bin (10) of 1,200 tons capacity. From this bin it goes to four 24-in. apron feeders (11), and thence via a horizontal 20-in. belt conveyor (12) to an inclined belt conveyor (13).

Conveyor (13) delivers the ore, ranging from $\frac{3}{4}$ -in. maximum size to the finest material coming from the mine, to two 54 by 18 in. Garfield rolls (14) and (16), which reduce it to 3-mm. maximum size. It then goes by way of No. 1 bucket elevator (15) to four Colorado Iron Works impact screens (17) with openings 2.5 mm. square. Oversize returns to rolls. Undersize goes via an eight-compartment mechanical distributor (20) to any number or all of eight No. 8 Wilfley roughing tables (21).

Principal Machines and Accessories in the Concentrator of the Timber Butte Milling Co., Shown on Flow Sheet, Figs. 3 and 3A.

Flow Sheet Number	Number of Pieces	Description of Machine
1	2	Crude-ore bins, 750 tons combined capacity.
2	1	Shaking-screen feeder, 1½-in. square openings.
3	1	15 by 30 in. Farrel jaw crusher, crushing to 3 in.
6	1	Steel elevator, 32-ft. centers—95 ft. per minute.
7	1	3 by 8 ft. Symons pulsating screen, 375 strokes per minute, ¾-in. round openings.
8	1	Symons 36-in., style C disk crusher, 133 and 300 rev. per minute, crushing to ¾ in.
9	1	24-in. inclined belt conveyor, 196 ft. per minute.
10	1	Secondary ore bin, 1,200 tons capacity.
11	4	24-in. steel apron feeders, 5 ft. per minute.
12	1	20-in. horizontal belt conveyor, 250 ft. per minute.
13	1	20-in. inclined belt conveyor, 250 ft. per minute.
14	1	54 by 18 in. Garfield rolls, 83 rev. per minute.
15	1	No. 1 belt elevator, 84-ft. centers, 415 ft. per minute.
16	1	54 by 18 in. Garfield rolls, 83 rev. per minute.
17	4	C.I.W. impact screens, 50 rev. per minute, 2½-mm. square openings.
20	1	Eight-compartment mechanical distributor, 21 rev. per minute.
21	8	No. 8 Wilfley roughing tables, 243 rev. per minute.
22	2	45-in. Akins classifiers, 5 rev. per minute.
23	1	No. 2 belt elevator, 72-ft. centers, 400 ft. per minute.
24	1	No. 3 belt elevator 50-ft. centers, 400 ft. per minute.
25	1	Dewatering box.
26	1	Three-compartment distributor.
27	3	8-ft. by 30-in. Hardinge mills, 28 rev. per minute.
28	1	No. 4 belt elevator, 58-ft. centers, 400 ft. per minute.
29	1	Eight-compartment mechanical distributor, 21 rev. per minute.
30	8	No. 3 James sand tables, 262 rev. per minute.
31	1	No. 5 belt elevator, 80-ft. centers, 400 ft. per minute.
32	1	Five-spigot Richards hindered-settling classifier, 40 tons capacity.
33	1	Double, three-compartment Harz jig.
34	2	No. 6 Wilfley tables, 240 rev. per minute.
35	1	No. 3 James sand table, 262 rev. per minute.
36	1	Distributing box.
37	2	45-in. Akins classifiers, 5 rev. per minute.
38	2	8-ft. by 30-in. Hardinge mills, 28 rev. per minute.
39	1	No. 6 belt elevator, 58-ft. centers, 490 ft. per minute.
40	1	16 by 10 ft. sludge tank, 10 rev. per minute.
43	1	Standard Minerals Separation 11-cell flotation machine, capacity 600 tons.
44	1	Standard Minerals Separation 8-cell flotation machine, capacity 200 tons.
45	1	Air lift.
48	1	4½-ft. Hardinge mill, 32 rev. per minute.
49	1	No. 8 belt elevator, 58-ft. centers, 490 ft. per minute.
50	3	45-in. Akins classifiers, 5 rev. per minute.
51	1	No. 7 belt elevator, 81-ft. centers, 490 ft. per minute.
52	1	Esperanza classifier, 25 ft. per minute.
53	1	Five-spigot Richards hindered-settling classifier, 100 tons capacity.
54	7	No. 3 James sand tables, 265 rev. per minute.
55	1	30-in. Akins classifier, 5 rev. per minute.
57	1	36-ft. Dorr thickener, ½ rev. per minute.
58	1	Cameron steam pump.
59	1	Cameron steam pump.
60	1	36-ft. Dorr thickener, ½ rev. per minute.
61	1	Two-stage centrifugal pump 1½ in.
62	1	10 by 10 ft. hot-water tank.
63	1	45-in. Akins classifier, 5 rev. per minute.
65	2	28-ft. Dorr thickeners, ½ rev. per minute.
66	1	No. 9 belt elevator, 52-ft. centers, 440 ft. per minute.
67	1	14 by 10 ft. agitating tank.
68	1	8 by 15½ ft. Montejú tank.
69	1	Kelly filter press, type 850.
70	1	Two-stage centrifugal pump.
71	2	Copper-concentrates bins.
72	2	Lead-concentrates bins.
73	2	Coarse-zinc concentrates bins.
74	2	Fine-zinc concentrates bins.
75	1	30,000-gal. dirty-water tank.
76	1	30,000-gal. fresh-water supply tank.
77	1	50,000-gal. sprinkler tank for fire protection.
78	2	Settling boxes in concentrate house.

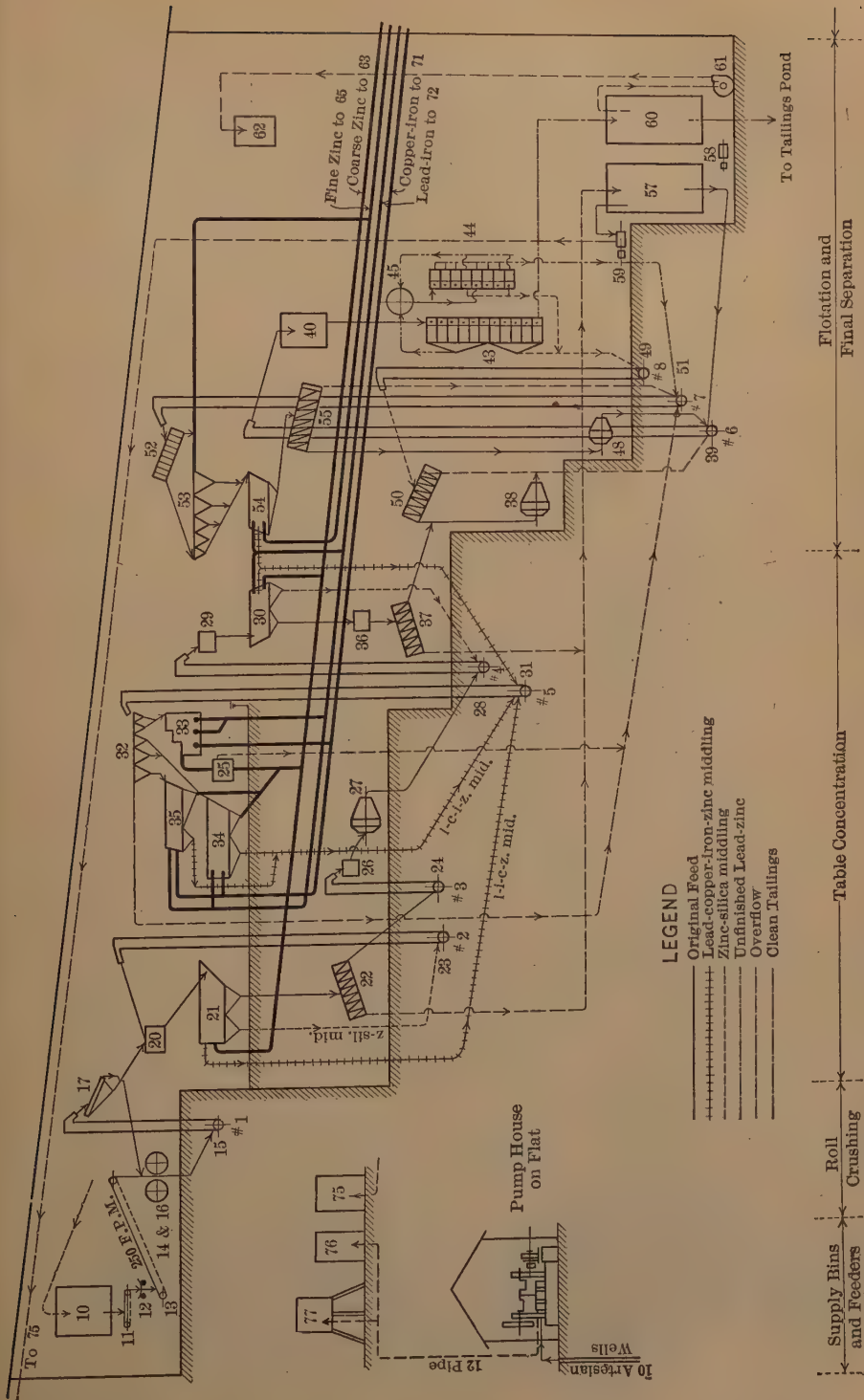


FIG. 3.—FLOW SHEET OF CONCENTRATOR. (See also Fig. 3A.) Numbers refer to list of machines, etc., on p. 916.

3. Table Concentration

Only four of these tables are used at present, each treating on an average 100 tons of crude ore per day, and also handling its proportion of circulating middlings. Each table makes the following products:

1. A lead-copper-iron-zinc middling, which goes to No. 5 elevator (31) for further treatment (see "Description of Processes").

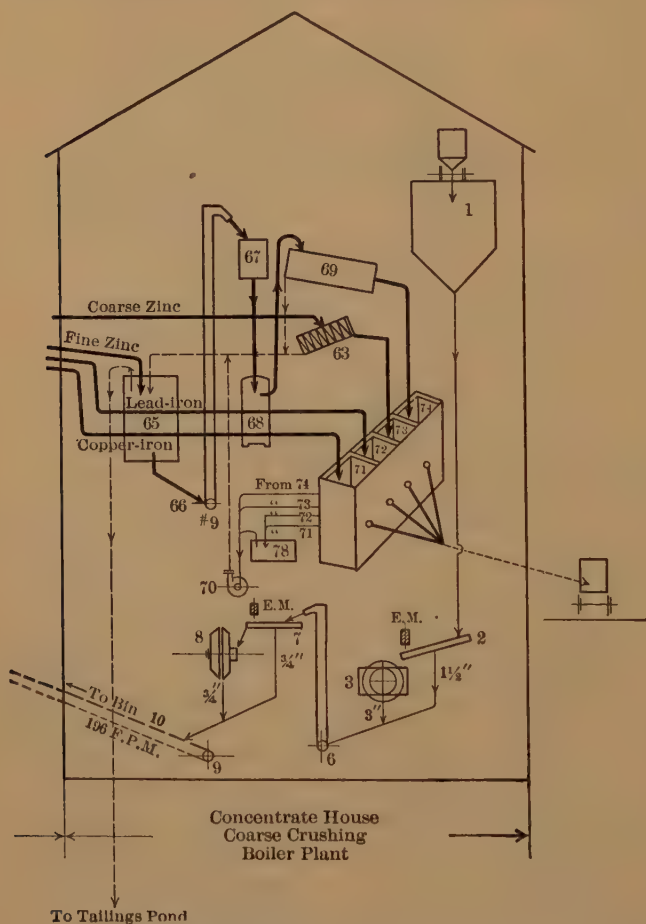


FIG. 3A.—FLOW SHEET OF CONCENTRATOR.

2. A coarse-zinc concentrate, which goes to bin (73) via classifier (63). In order to keep this product at all times as free as possible from silica it is cut off at a safe point near the lower end of the concentrates discharge. This allows some of the free zinc to go over into the next product.

3. A zinc-silica middling—so called—which is returned to the same tables via No. 2 elevator (23) and distributor (20). In this manner the free zinc that was permitted to mix with this product eventually finds its way into the zone in which it belongs, while the true middling particles join the tailings product No. 4 to go to Hardinge mills (27) for regrinding.

4. Tailings and slimes, which pass to two 45-in. Akins classifiers (22) for further treatment.

Product No. 1 is delivered by No. 5 elevator (31) to a five-spigot Richards hindered-settling classifier (32), the overflow from which goes to No. 7 elevator (51). The first two spigots of the classifier discharge into a double three-compartment Harz jig (33). Only one side is used at present. The jig makes the following products:

1. A finished lead concentrate from cup and hutch of first two compartments, which goes to bin (72).

2. A finished copper-iron concentrate from cup and hutch of third compartment, which goes to bin (71).

3. The tail overflow is a coarse-zinc concentrate which, after being partly dewatered in box (25), goes to bin (73) via classifier (63). The overflow of dewatering box (25) goes to No. 7 elevator (51).

The third and fourth spigot product of classifier (32) goes to two No. 6 Wilfley tables (34); the fifth spigot feeds one No. 3 James sand table (35). Wilfleys and James table each make four products:

1. A lead-iron concentrate, which goes to bin (72).

2. A copper-iron concentrate, which goes to bin (71).

3. A copper-iron-zinc middling—so called—which is returned to same tables via No. 5 elevator (31) and classifier (32).

4. Tailing, which is a coarse-zinc concentrate, and goes to bin (73) via classifier (63).

Additional tables are being installed for this work, which will result in a better separation of the minerals.

Tailings from the Wilfley roughing tables (21) go to two 45-in. Akins classifiers (22), which overflow to a 36-ft. Dorr thickener (57). Sands go via No. 3 elevator (24) to a three-compartment distributor (26) which feeds three 8-ft. by 30-in. Hardinge mills (27). Only two of them are used at any one time, permitting repairs of the third one. Mills furnish a product of approximately 20-mesh maximum size. This goes via No. 4 elevator (28) and mechanical distributor (29) to all or any of eight No. 3 James sand tables (30). Only five of these are used at present; and each makes the following products:

1. A lead concentrate, which goes to bin (72).

2. A lead-copper-iron-zinc middling—so called—which goes to No. 5 elevator (31) for re-treatment on jig (33), Wilfleys (34), and James table (35).

3. A coarse-zinc concentrate, which goes to bin (73) via classifier (63).

4. A zinc-silica middling, which contains some of the free zinc of the preceding product, and is therefore returned to the same tables via No. 4 elevator (28) for the reasons explained for the Wilfley roughers (21).

5. Tailings, which go via distributing box (36) to two 45-in. Akins classifiers (37). Overflow goes to Dorr thickener (57). Sands to two 8-ft. by 30-in. Hardinge mills (38), where it is attempted to crush to a maximum size of 60 mesh. Product is delivered by No. 6 elevator (39) to a 16 by 10 ft. sludge tank (40) in the flotation section.

4. Flotation Section and Final Separation

In the flotation section, the final separation of the fine sulphide minerals from the gangue takes place.

For a satisfactory separation acid (H_2SO_4) and heat are found necessary adjuncts to the generally accepted oil-flotation principles.

From the sludge tank (40), which acts as an equalizer between the gravity-concentration and flotation section, the pulp flows to a standard 600-ton Minerals Separation unit (43) known as the "rougher machine." It consists of eleven cells, each divided into a mixing or agitating and a flotation compartment. The intensely agitated pulp discharges from the mixing into the flotation compartment, where the pulp is comparatively still and where a portion of the sulphide minerals rises to the surface to be skimmed off by revolving paddles and discharged into a launder in front of the cells. The remainder of the sulphides and the gangue that has settled to the bottom of each flotation compartment are drawn into the adjoining cell by the pumping action of its agitator. Here the process is repeated and sulphides not floated in the first cell are given a chance in the second cell, and so on.

The overflow from the last five cells of the "rougher machine" (43) is a middling that goes via No. 8 elevator (49) to Akins classifiers (50). The classifier overflow returns to the "rougher machine" of the flotation section. Sands are reground in two 8-ft. by 30-in. Hardinge mills (38) to an approximate 60-mesh maximum size and return via No. 6 elevator (39) to flotation section.

Finished tailings from the eleventh cell of the "rougher machine" (43) flow to a 36-ft. Dorr thickener (60) to be partly dewatered and thence to the tailings pond. The overflow from this tank is hot water, free from solids. It is pumped by a two-stage centrifugal pump (61) to a 10 by 10 ft. tank (62), which supplies the flotation machines with hot water.

The mixed sulphide concentrate which overflows from the first six cells of the "rougher machine" (43) goes via air lift (45) to a 200-ton Minerals Separation unit (44), consisting of eight cells, known as the "finisher." Here the "rougher" concentrates are refloatated for the pur-

pose of removing more of the gangue minerals. Cells Nos. 1-2-3-5-6-7 of this machine (44) make a float sulphide which goes via No. 7 elevator (51) to classifiers and James tables for final separation of lead and iron from the zinc.

Overflow from cells 4 and 8 of "finisher" (44) returns via air lift (45) to the same unit. The bottom discharge from cells 4 and 8 is a middling which joins the middlings from the "rougher machine" (43) to be reground in Hardinge mills (38) and returned to the flotation section.

The float concentrates from unit (44) are lifted by No. 7 elevator (51) to an Esperanza classifier (52), the overflow from which is a finished fine-zinc concentrate that goes to two 28-ft. Dorr thickeners (65) in the concentrate building. Sands go to a five-spigot Richards hindered-settling classifier (53), whose overflow is a finished fine-zinc concentrate, which joins the overflow of the Esperanza classifier.

The discharge from the five spigots of the Richards classifier is distributed on seven No. 3 James sand tables (54), where the lead and zinc are separated and the insolubles in the zinc concentrate reduced to about 3.5 per cent. Each table makes the following products:

1. A finished lead concentrate, which goes to the bin (72).
2. A lead-copper-iron-zinc middling, which is returned via No. 5 elevator (31) to the primary jig and table re-treatment circuit.
3. A fine-zinc concentrate, which goes to Dorr thickeners (65) in the concentrate building to be partly dewatered.
4. A tailing, which is reground in a $4\frac{1}{2}$ -ft. Hardinge mill (48) after passing through a 30-in. Akins classifier (55) whose overflow returns via No. 7 elevator (51) to the final table-separation circuit.

The product of the Hardinge mill (48), which is of about 60-mesh maximum size, goes to flotation section via No. 6 elevator (39).

The Dorr thickener (57) receives practically all the fines and slimes coming from the mine and those made in the crushing operations. The overflow is dirty water, which is pumped by a double Cameron steam pump (59) to the "dirty-water tank" (75) at the top of the mill, whence it returns for circulation throughout the mill. A duplicate auxiliary Cameron pump (58) is put into service whenever pump (59) is stopped for repairs. From the bottom of tank (57) the thickened pulp is lifted by hydrostatic pressure to the boot of No. 6 elevator (39) to go to flotation section.

5. Concentrate House

The chief operation carried on in this department is the dewatering of the fine-zinc concentrates made in the flotation and final separation section. These concentrates are all collected in two 28-ft. Dorr thickeners (65), the overflow from which is practically clear water. This goes at present to waste, joining the tailings from tank (60) in the flota-

tion section. The thickened concentrates, carrying about 30 per cent. moisture, are discharged at the bottom of the Dorr thickeners (65) and go via No. 9 elevator (66) to a 14 by 10 ft. agitating tank (67), and from there to an 8 by 15½ ft. Montaju tank (68) that carries 80 pounds air pressure. From this tank they are forced into a Kelly filter press (69), 850 type, where the moisture is reduced from 30 to about 10 per cent. The liquid from the press is practically clear water and returns to the Dorr thickeners (65), while the cake drops into the fine-zinc concentrate bins (74) whence it is drawn into the railroad cars for shipment to the smelter.

The coarse-zinc concentrates upon entering this building pass through a 45-in. Akins classifier (63) to be dewatered, before going to the coarse-zinc concentrate bins (73). The overflow from classifier (63) returns to the Dorr thickeners (65).

The copper-concentrate bins (71) and the lead-concentrate bins (72) are each provided with a filter bottom through which much of the moisture in the concentrates is drained and passed to two settling tanks (78), the overflow from which is pumped back to the Dorr thickeners (65) by a two-stage centrifugal pump (70).

The coarse (73) and fine zinc (74) concentrate bins also have filter bottoms and drain, whenever there is sufficient moisture in bin products, direct to the sump of centrifugal pump (70).

III. OPERATING DETAILS

1. *Capacity, Efficiency, and Economy of Mill*

Since operations began in June, 1914, the quantity of crude ore treated has been increased from 350 to 450 tons per day. The ratio of concentration of crude ore into zinc concentrates in round figures is three into one. More exactly, for the month of September, 1914, it was 2.89 into one. At this ratio 450 tons of ore treated per day would result in 150 tons of zinc concentrates.

Roughly, 74 per cent. of this amount, or 111 tons, is fine zinc, including concentrates from flotation, final separation, and the fine portions of the table concentrates.

The ratio of concentration of crude ore into lead and copper-iron products is approximately 125 into one, 20 per cent. of which is lead concentrates.

Treatment costs have averaged between \$2 and \$2.25 per ton of ore passed through the mill. This covers labor, power, water, supplies, repairs, and general expenses, but not interest on investment and depreciation.

TABLE I.—Average Analysis of Crude Ore, Concentrates, and Tailings

Name of Product	Cu, Per Cent.	Ag, Oz. per Ton	Au, Oz. per Ton	Fe, Per Cent.	Zn, Per Cent.	Pb, Per Cent.	Insol., Per Cent.
Original ore:.....	0.73	6.23	0.013	3.29	18.52	0.75	62.23
Copper-iron concentrates	2.25	21.01	0.04	19.78	22.78	13.01	2.30
Lead-iron concentrates..	1.13	18.91	0.04	9.99	9.15	50.60	5.05
Coarse-zinc concentrates	1.96	14.74	0.035	9.15	47.77	1.88	3.47
Fine-zinc concentrates..	2.22	16.84	0.036	4.45	52.44	1.72	5.05
Tailings.....	0.07	0.60	0.0017	0.44	1.20	0.05	95.42

A comparison of the tonnage of concentrates and their analysis with the tonnage and analysis of the crude ore milled, shows an extraction of about 96 per cent. of the zinc and about 92.6 per cent. of the silver contents of the original ore.

The copper-iron concentrates go to the Washoe works at Anaconda, Mont.; the lead concentrates to the lead smelter of the United States Smelting Co. at Midvale, Utah; and the zinc concentrates to the zinc smelters at Bartlesville, Okla., and Argentine, Kan.

2. Description of Processes

A. Gravity-Concentration Section.—The most striking feature in this section of the mill is the performance of the Wilfley roughing tables. They are the first concentrating machines encountered in the mill and take the undersize from the impact screens without any previous classification. This material ranges from 2.5 mm. maximum diameter to the finest stuff in the ore. The tables not only handle an unusual quantity of material, amounting to more than 100 tons per table daily, but they make at the very outset a finished coarse-zinc concentrate, containing from 49 to 51 per cent. zinc and not more than 4.0 per cent. insolubles. Moreover, they also send all the fines coming from the mine and those made in the preceding crushing operations to two Akins classifiers to be separated from the sands, after which they go directly to the M. S. flotation section where they belong. Thus none of the intermediate machines are incumbered with this troublesome material, and the losses incident to frequent handling of slimes are reduced to a minimum. Moreover, the tables furnish as their first product a so-called lead-copper-iron-zinc middling which by the elimination of most of the silica has been enriched to a point where it can be profitably re-treated.

This product No. 1 is taken from the portion of the tables where the heavier-than-blende sulphides collect. With the Elm Orlu ore this results in a concentration of the lead-iron and copper sulphides at this point. The "cut" of product No. 1 is made at a point which will not only prevent excessive amounts of the mixed lead-copper-iron sulphides

from entering and contaminating the next following product No. 2, intended as a high-grade finished zinc concentrate, but will also prevent excessive amounts of free zinc blende in product No. 1.

Product No. 1 is virtually a concentrate in which the insoluble has been reduced to the economic limit. It might be called a "combination of concentrates," being in reality a mixture of sulphides of the base metals. As such it has no commercial value; and, pending further treatment, it is considered a "middling" in the sense of being an unfinished product.

Considering the zinc blende in this product as "the" sulphide and the other sulphides as the gangue, all conditions for a true middling would prevail; *i.e.*, a mechanical mixture of the finer, already free sulphides up to the coarser sizes, consisting of the metallic sulphides interlocked with gangue minerals. In the case under consideration, the interlocked minerals are in amounts too small to warrant regrinding, an operation usually associated with the conception of a middling product.

From the original mill feed, containing only a small percentage of lead and copper, it is practically impossible to make commercial products of those metals by gravity methods in one operation, without a large installation of machinery and without high treatment charges. However, by the roughing operations it is possible to make this rough middling product No. 1, which, due to the enrichment and to the relatively small tonnage of the product, becomes amenable to final economical separation by gravity processes. Thus the taking off of the No. 1 product from the roughing tables results: first, in a much higher grade of finished zinc concentrate by the exclusion of other sulphides; and second, in making possible the economic gravity separation of these sulphides into a finished lead concentrate and a finished copper-iron concentrate.

All this is accomplished by a clever modification and distribution of the riffles, by giving the tables a rather steep inclination (8° to 10°), and by adding the proper quantity of wash water.

The remarks made here about this product No. 1 apply with equal force to the lead-copper-iron-zinc middling product made on the James roughing tables (30).

B. "*Minerals Separation*" *Flotation Section*.—Flotation processes like the one employed in this mill present such unusual possibilities in their adaptation to the treatment of metalliferous ores that the study of a process which has successfully solved the problem of treating a particular ore is of the highest interest. Although, in America, flotation has so far been most successfully employed in the treatment of sulphide ores, it seems highly probable that when the interrelation of the underlying chemical and physical laws is more fully understood flotation will be extended with equal success to the treatment of oxide and carbonate ores.

The simplicity of the processes and of the machinery employed; the small floor space occupied by such machinery compared with old-type concentrators of the same capacity; the fact that they accomplish a separation where other processes have failed; and their many other advantages, are so striking that it looks to be merely a question of time when flotation will not only usurp many of the functions of the older gravity processes but will eventually supplant them altogether, at least for the treatment of the finer material and of slimes.

In view of the scientific and commercial importance of flotation, a brief outline of the plant installed at the Timber Butte mill is given in the following pages:

To unlock the finest included grains of sulphides from the adhering gangue, prior to separation, it is necessary to crush the Elm Orlu ore to a size between 40 and 60 mesh maximum. In the course of tests and experiments, preceding the design and building of the mill, it was found that attempts at extracting the zinc from material finer than 40 mesh by a simple process of gravity concentration, resulted in a low grade of concentrates, which could only be increased by a costly installation.

This led to the adoption of "flotation" according to the process of the Minerals Separation Co. In this process use is made of the now well-known facts that when an ore containing metallic sulphides is mixed with water, oil, and acid and agitated together in proper proportions, the oil not only has a selective action for the sulphides, but the air bubbles formed by agitation have a selective action for the oiled sulphide particles, to which they attach themselves and thus float them to the surface in the shape of a froth which overflows or is mechanically skimmed off. The gangue minerals, on the other hand, have a comparatively feeble adhesiveness to gas films and oil, and a strong adhesiveness to water, which is greatly increased by a slight acidulation of the water. For this reason they quickly become wetted and sink to the bottom of the flotation machine, resulting in the desired separation of the sulphides from the gangue.

No satisfactory theory has as yet been propounded, as to why acid promotes the preferential adhesion of water to gangue and also probably increases at the same time the preferential adhesion of oil to sulphides. For commercial purposes it suffices to recognize the facts and make practical use of them.

This is done in the Minerals Separation process by passing the mixed pulp through one or more standard machines, consisting of a series of cells which have the shape outlined in Fig. 4. At the Timber Butte mill, the pulp is kept at a uniform density in the proportion of three of water to one of ore, and is heated to a temperature of 130° F. by the introduction of live steam. From the tests it appeared that with the Elm Orlu

ore higher temperatures did not produce better results, while temperatures below 125° F. reduced the efficiency of the plant.

One of the effects of heating the pulp is the minimizing of the force of adhesion which tends to surround the mineral particles with water; that is, to wet them and thereby detach the air bubbles that are instrumental in floating them to the surface.

In the Timber Butte mill, sulphuric acid is mixed with the pulp at the rate of 7 to 8 lb. of acid to every ton of original ore going through the mill. Oil is added in the proportion of 0.5 lb. per ton. The pulp is first passed through a standard 600-ton Minerals Separation unit (43) consisting of eleven cells. This is known as the "rougher machine," because it is used, not to make a finished product, but rather to treat a maximum amount of material "roughly;" that is, to get rid of all clean tailings and save for re-treatment all the sulphides and the more or less

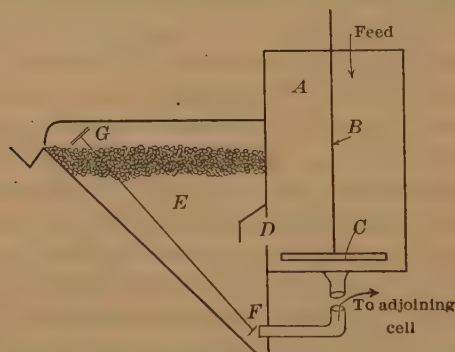


FIG. 4.—OUTLINE OF M. S. FLOTATION CELL.

rich middlings. At the Timber Butte mill, oil is introduced in the mixing compartment (A) of the first cell where the pulp is violently churned by the action of the agitator (B), which at this plant makes 265 rev. per minute and causes the propellers (C) to run at a peripheral speed of about 1,500 ft. per minute. The aerated mixture passes through the opening (D) into the flotation compartment (E), where the froth containing the sulphides of lead, copper, zinc, and iron, as well as some of the richer middlings, immediately rises to the surface, while the clean gangue and the leaner middlings drop to the bottom.

The flotation compartment has the shape of a spitzkasten, the apex of which is provided with a valve (F), adjustable by a hand wheel (G). Through this valve the material that has settled to the bottom of the first cell is drawn into the adjoining cell by the pumping action of its propeller. The process is here repeated and sulphides which did not float in the first cell are given a chance to do so in the second one, and so on. By the time the pulp has reached the seventh cell of the "rougher

machine" the clean sulphides and the richer middlings have all been floated. The remaining middlings and tailings in their passage from cell No. 7 to cell No. 11 are separated from each other by the successive floating of the middlings until in the bottom of the last cell (No. 11) only clean tailings remain, which are taken to a Dorr thickener (60) for extraction of the hot water. This is pumped to a tank (62) to be re-used in the flotation machines.

The middlings overflowing from cells 7 to 11 are reground in Hardinge mills (38) and returned to the flotation section.

The float concentrate from the first six cells of the "rougher machine" (43) is a product rich in zinc (about 47.8 per cent.), but due to the presence of middlings still contains about 14 per cent. insolubles. It is therefore sent to the first and fifth cells of a 200-ton Minerals Separation flotation unit (44) containing eight cells. This is known as the "finisher machine" because it makes a finished sulphide concentrate. On passing the pulp through the successive cells of the "finisher," the sulphides are floated to the surface while the clean middlings ultimately reach the bottom of the fourth and eighth cells; whence they go to Hardinge mills (38) to be reground and returned to the flotation section.

The float concentrate from cells 1-2-3-5-6-7 of the "finisher machine" (44) is a finished sulphide product containing, besides the sulphides of lead, copper, and iron, 52 per cent. zinc and 3.5 per cent. insolubles. The fines in this product overflow as fine-zinc concentrate from an Esperanza and a Richards classifier through which the product is passed, the coarser portion being re-treated on James tables (54). On these tables the lead and iron are separated from the zinc and the insolubles in the zinc concentrate are reduced to 3.5 per cent.

The overflow from cells 4 and 8 of the "finisher machine," being apt to contain a varying amount of middlings, is returned via air lift (45) to the same machine.

The agitating compartments of the flotation cells are lined with cast iron, and the propellers are made of brass. All agitators of a standard machine are driven from one main shaft by gears which are protected from dust by steel housings. Gears are placed so that propellers in adjoining cells revolve in opposite directions, thus equalizing the strains on the machine and the power transmission.

IV. POWER

1. *Electric Power*

Electric power is furnished over a 2-mile transmission line from the substation of the Montana Power Co. on Montana Street near the old Butte reduction works. Three Westinghouse 350 K.V.A. transformers,

housed in a fireproof concrete building, reduce the voltage from 3,800 to 460 volts for use in the mill motors.

Individual motors are used to drive individual machines or sets of machines.

List of Electric Motors

Motor No.	Horse-power	Machinery Driven by Motors	Machine's Number on Flow Sheet
1	150	2 sets of Garfield rolls.....	14 and 16
		4 apron feeders.....	11
		2 belt conveyors.....	12 and 13
		No. 1 elevator.....	15
		4 impact screens.....	17
2	50	8 Wilfley roughing tables.....	21
		No. 4 elevator.....	28
		No. 5 elevator.....	31
		Double three-compartment jig.....	33
		2 Wilfleys.....	34
		1 James table.....	35
		1 distributor.....	20
3	20	No. 2 elevator.....	23
4	50	1 Hardinge mill.....	27
5	50	1 Hardinge mill.....	27
6	50	1 Hardinge mill.....	27
7	7½	2 Akins 45-in. classifiers.....	22
8	7½	2 Akins 45-in. classifiers.....	37
9	20	No. 3 elevator.....	24
10	40	8 James roughing tables.....	30
		7 James finishing tables.....	54
		No. 7 elevator.....	51
		Esperanza classifier.....	52
		Mechanical distributor.....	29
11	50	1 Hardinge mill.....	38
12	50	1 Hardinge mill.....	38
13	30	No. 6 elevator.....	39
14	30	No. 8 elevator.....	49
15	20	1 Hardinge mill.....	48
		3 Akins 45-in. classifiers.....	50
		1 Akins classifier.....	55
16	75	Rougher flotation cells.....	43
17	25	Finisher flotation cells.....	44
18	7½	2 Dorr thickeners and sludge tank.....	57, 60, and 40
19	25	1 double centrifugal pump.....	61

List of Electric Motors (Continued)

Motor No.	Horse-power	Machinery Driven by Motors	Machine's Number on Flow Sheet
20	125	1 shaking screen.....	2
		1 Farrel crusher.....	3
		1 Symons screen.....	7
		1 Symons crusher.....	8
		1 steel elevator.....	6
		1 Ingersoll two-stage compressor.....	
21	25	2 Dorr thickeners.....	65
		1 Akins classifier.....	63
		No. 9 elevator.....	66
22	20	1 centrifugal pump.....	70
23	20	Belt conveyor, lower section.....	9
24	20	Belt conveyor, upper section.....	9
25	50	Clayton two-stage compressor.....	
26	37	Tramway hoist.....	
27	20	Vacuum pump in pumping station.....	
28	150	Triplex pump.....	

A number of smaller motors are employed in and about the mill for grinding samples, for running tools in the shops, etc.

From meter readings, the consumption of electric power during the month of October, 1914, was as follows: In the mill, 803.6 hp., or 1.99 hp. per ton of ore milled. At the pumping station, 120.2 hp.

2. Steam Power

Steam required in the flotation section for heating the pulp and for running the Cameron pumps is furnished by two 200-hp. Springfield boilers, while one 150-hp. boiler supplies steam for the radiators in the mill and other buildings. The boilers are all connected with each other and the steam is carried to the various buildings in concrete underground conduits.

3. Compressed-Air Power

An Ingersoll-Rand, Imperial type, two-stage compressor with a capacity of 240 cu. ft. of free air per minute, and a Clayton two-stage compressor with a capacity of 300 cu. ft. per minute, supply air for operating the air gates at the ore bins and the Kelly filter press; for pumping the acid from the railroad tanks into the mill tank; for various tools in the repair shops, and for the air lift in the flotation section.

V. WATER SUPPLY

Fresh water for milling purposes is pumped to a 30,000-gal. tank (76) at the top of the mill from the company's pumping station on the Flat, about 2 miles northeast of the mill. At this plant 10 artesian wells, ranging between 38 and 59 ft. in depth, are connected on top to a duplex 12 by 12 in. vacuum pump, driven by a 20-hp. electric motor. The vacuum pump delivers the water into a sump underneath a Platt Iron Works triplex 11 by 15 in. pump of 750 gal. capacity per minute, driven by a 150-hp. electric motor. This pump raises the water in one lift through a 12-in. pipe to the mill tank (76), 450 ft. above the pump station, whence it flows by gravity to the various sections of the mill. Fresh water is used for wash water on Wilfley and James tables, also in jig and in two Richards hydraulic classifiers.

Clear water for future requirements is also available from the overflow of the two Dorr thickeners (65) in the concentrate house. This overflow, which amounts to about 530 gal. per minute, at present goes to the tailings pond. In case of failure of the pumping plant on the Flat to do its work, the fire pump at the mill can be started up to supply the mill tank with city water, to which the pump is connected.

Dirty water, amounting to about 130 gal. per minute, overflows from the Dorr thickener (57) in the flotation section. It contains 0.9 per cent. solids and is pumped by a Cameron steam pump to the dirty-water tank (75) at the top of the mill, whence it returns by gravity to the various sections of the mill. This water is used in rolls, on impact screens, on the feed end of Wilfley and James tables, in Hardinge mills, etc.

Hot water overflows from the Dorr thickener (60) at a variable rate. It is used in the flotation machines (43 and 44), being supplied from a hot-water tank (62), to which it is pumped by a centrifugal pump (61.)

VI. FIRE PROTECTION

A complete system of fire protection has been installed throughout the mill and surrounding buildings. It consists of the automatic sprinkler system of the International Company of Philadelphia, and is supplied with water from a 50,000-gal. tank (77) at the top of the mill. This tank is kept filled and used only for fire purposes. A set of steam coils keeps the water from freezing in cold weather. A by-pass in the main pipe line supplying fresh water to the mill tank (76) from the pumping station on the Flat, also supplies the sprinkler tank (77), which is located at an elevation high enough to send a stream of water by hydrostatic pressure to the highest point of the mill. Thirteen hydrants are scattered over the premises, in convenient location to cover all portions of the plant (see Fig. 2).

If for any reason the water supply from the pumping station drops below a certain limit and pressure, a large fire pump is automatically started which supplies the fire circuit with water from the system of the Butte Water Works.

The fire pump is a triplex 11 by 15 in. Platt Iron Works pump, driven by a 125-hp. electric motor and having a capacity of 750 gal. per minute against 200 lb. pressure. The pump is housed in a concrete, fireproof building, at an elevation which permits the sump underneath it to fill with city water by hydrostatic pressure as fast as it is taken out.

VII. CONCLUSIONS

A visitor to the Timber Butte mill cannot help being impressed at once by the up-to-date appearance of the plant and the completeness with which every detail has been worked out that makes for efficiency, economy, and comfort. There is room in abundance for all machinery. Every part of it is accessible and illuminated in daytime by a wealth of natural light and at night by a liberal distribution of large tungsten lamps.

Shafts and belts are arranged so as to minimize the danger to attendants and at the same time be easily accessible for lubricating and necessary repairs. Launderers, elevators, and conveyors are systematically arranged and open for inspection throughout their whole length.

Numerous steel radiators keep the mill comfortably warm in cold weather and a large fan in the flotation section frees the atmosphere from excessive moisture due to condensation of steam.

A spacious modern office building and a large assay office supplied with all modern conveniences add materially to the completeness and appearance of the whole plant, every detail of which manifests the foresight of the company and the skill of its engineers.

Hardinge Mill Data

BY ARTHUR F. TAGGART,* NEW HAVEN, CONN.

(San Francisco Meeting, September, 1915)

THE following conclusions on the work of the Hardinge mill are based on data furnished to the writer by the Hardinge Conical Mill Co. in the form of the mesh cards hereto appended. Energy units (E. U.) and relative mechanical efficiencies (R. M. E.) are computed by the "volume method" of Stadler.¹ Screen apertures used are the average apertures of testing screens of the meshes given.

Card 122. June 28, 1912. Vipond Porcupine Mines Co., Ltd., Schumacher, Ont., Canada.

Ore from mill bin. Gangue, quartz and basalt.

4.5 ft. by 13 in. ball mill.

Capacity, 48 tons per 24 hr.

Charge, 4,000 lb. balls.

Speed, 33 rev. per minute.

Horsepower, 15 to 17.

Water, 100 per cent. by weight. (50 per cent. ?)

Product deslimed and oversize reground in pebble mill, see Card 113.

Feed to mill through 2-in. mesh.

Mesh	Feed, Per Cent.	Discharge, Per Cent.
+ 1 in.	5.50
+ $\frac{3}{4}$ in.	28.00
+ $\frac{1}{2}$ in.	30.00
+ $\frac{1}{4}$ in.	19.72
+ 10	10.87	2.10
+ 20	2.42	8.00
- 20	3.51
+ 40	22.68
+ 60	10.50
+ 80	11.80
+ 100	3.90
- 100	40.15

* Sheffield Scientific School.

¹ H. Stadler: Grading Analyses and Their Application, *Transactions of the Institution of Mining and Metallurgy*, vol. xix, p. 471 (1910-11).

Arthur F. Taggart: The Work of Crushing, *Trans.*, xlviii, 153 (1914).

Card 107. Jan. 30, 1912. Miami Copper Co., Miami, Ariz.

Material from mill bin. Gangue, siliceous porphyry.

6 ft. by 16 in. ball mill.

Capacity, 351 tons per 24 hr.

Charge, 4 tons balls.

Speed, 28 rev. per minute.

Horsepower, 35 net.

Water, 50 per cent. (approx.).

Elevation of feed end, 2 in.

Consumption of balls, 0.578 lb. per ton of ore crushed.

Feed to mill through 1.5-in. mesh.

Mesh	Feed, Per Cent.	Discharge, Per Cent.
+ ¾ in.	6.7
+ ½ in.	19.7
+ ¼ in.	28.5	5.2
+ 10	22.3	16.3
+ 20	7.5	20.3
+ 40	3.0	11.9
+ 50	1.1	4.9
+ 60	1.6	6.8
+ 80	0.8	3.4
+100	0.8	2.8
+150	0.9	3.1
+200	1.0	3.5
-200	7.2	21.8

Card 155. Aug. 3, 1914. Britannia Mining & Smelting Co., Britannia Beach, B. C., Canada.

Jig tailing. Gangue quartzose, very hard.

6 ft. by 16 in. ball mill.

Capacity, 251 tons per 24 hr., average of six tests.

Charge, 8,200 lb. of 2-in. cast-iron balls.

Speed, 28 rev. per minute.

Horsepower, 38 to 40.

Water, 40 per cent.

Elevation of feed end, 0

Consumption of balls, 0.72 lb. per ton of ore.

Lining in first-class condition after 3 months' run.

This mill is taking "pebble mill feed" (all through ¼-in. aperture), but using small balls instead of pebbles as a grinding medium. These data are, therefore, not included in the averages in Table II.

Mesh	Feed, Per Cent.	Discharge, Per Cent.
+ 10	60.4	0.4
+ 20	28.8	5.5
+ 30	5.8	10.9
+ 40	1.1	6.0
+ 60	1.9	18.0
+100	1.2	23.3
-100	1.1	34.2

Card 192. Nov. 17, 1914. McIntyre Porcupine Mines, Schumacher,
Ont., Canada.

Quartz and schist.

6 ft. by 16 in. ball mill.

Capacity, 150 tons per 24 hr.

Charge, 8,000 lb. of balls.

Speed, 28 rev. per minute.

Horsepower, 36.

Water, 50 per cent.

Elevation of feed end, 1.25 in.

Consumption of balls, 0.5 lb. per ton of ore ground.

Feed to mill through 2-in. mesh.

Mesh	Feed, Per Cent.	Discharge, Per Cent.
+ 1.5 in.	6.18
+ 1 in.	16.62
+ ½ in.	34.36
+ ¼ in.	22.90
+ 10	19.94	0.70
+ 20	5.49
+ 40	18.46
+ 60	11.61
+100	10.83
+200	11.84
-200	41.07

Card 156. June 25, 1914. McIntyre Porcupine Mines, Schumacher,
Ont., Canada.

Quartz and schist, hard.

6 ft. by 16 in. ball mill.

Capacity, 150 tons per 24 hr., average of nine months.

Charge, 4 tons of balls.

Speed, 28 rev. per minute.

Horsepower, 36.

Water, 1.5 tons KCN solution to 1 ton of dry ore.

Elevation of feed end, 1.25 in.

Consumption of balls, 0.5 lb. per ton of ore.

Feed to mill through 2-in. mesh.

Mesh	Feed, Per Cent.	Discharge, Per Cent.
+ 2	73.4
+ 4	12.2
+ 10	7.3
+ 20	2.4	15.6
+ 40	0.5	19.2
+ 60	1.2
+ 80	15.2
+100	0.6	3.8
+200	0.7	9.0
-200	1.7	37.2

Card 191. Sept. 14, 1914. Buckhorn Mines Co., Buckhorn, Nev.
 Soft talcose gold ore. Gangue, decomposed porphyry and basalt.
 6 ft. by 16 in. ball mill.
 Capacity, 160 tons per 24 hr.
 Charge, 8,000 lb. of balls.
 Speed, 28 rev. per minute.
 Horsepower, 33.25 input to motor.
 Water, 4 to 1.
 Elevation of feed end, 1.5 in.
 Consumption of balls, 0.45 lb. per ton of ore.
 Feed to mill through 1.5-in. mesh.

Mesh	Feed, Per Cent.	Discharge, Per Cent.
+ ½ in.	35.2
+ ⅜ in.	12.2
+ ¼ in.	13.2
+ ⅛ in.	13.8
+ 20	12.2	3.0
- 20	13.4
+ 40	12.0
+ 80	31.0
+100	10.0
+150	16.0
-150	28.0

Card 121. Bunker Hill & Sullivan Mining & Concentrating Co., Kellogg, Idaho.

Middling from 3-mm. jigs and tables. Gangue, quartzite and siderite.

6 ft. by 22 in. pebble mill.

Capacity, 60.3 tons per 24 hr.

Speed, 32 rev. per minute.

Horsepower, 16 (?).

Water, not taken. Previous test gave 75 per cent.

Elevation of feed end, 0.5 in.

Pebble load, 4,000 lb. (?).

Feed to mill through 3-mm. mesh.

Mesh	Feed, Per Cent.	Discharge, Per Cent.
+ 10	3.0
+ 20	50.7
+ 40	21.8
+ 60	3.5	2.8
+ 80	2.5	13.0
+100	4.5	14.1
+150	5.4	15.8
+200	4.9	12.5
-200	3.7	41.8

Card 113. June 28, 1912. Vipond Porcupine Mines Co., Ltd., Schumacher, Ont., Canada.

Oversize of Colbath classifier. Gangue, quartz and basalt.

6 ft. by 72 in. pebble mill.

Capacity, 40 tons per 24 hr.

Charge, 9,000 lb. of pebbles.

Speed, 27 rev. per minute.

Horsepower, 30.

Water, 50 per cent.

Elevation of feed end, 0.

Product deslimed and oversize returned.

Mesh	Feed, Per Cent.	Discharge, Per Cent.
+ 10	16.64
+ 20	27.42
+ 40	27.04
+ 60	8.88
+ 80	7.04	0.15
+100	5.00	1.95
+150	5.05
+200	1.95	29.10
-200	68.80

Card 108. Jan. 30, 1912. Miami Copper Co., Miami, Ariz.

Product from 16 by 42 in. rolls. Gangue, altered schist.

8 ft. by 22 in. pebble mill.

Capacity, 101 tons per 24 hr.

Charge, 10,000 lb. of pebbles.

Speed, 27 rev. per minute.

Horsepower, 36.

Water, 63 per cent.

Elevation of feed end, 1.5 in.

Product desired to pass 30 mesh with a minimum of slime.

Feed through $\frac{1}{2}$ -in. mesh.

Mesh	Feed, Per Cent.	Discharge, Per Cent.
+ 4	10.7
+ 10	28.8
+ 20	37.0	3.4
+ 30	11.4	9.2
+ 40	1.8	7.3
+ 60	1.5	13.9
+ 80	0.4	7.8
+100	0.6	7.2
+150	0.5	7.3
+200	0.7	8.0
-200	6.6	35.9

Card 80. Aug. 16, 1911. Miami Copper Co., Miami, Ariz.
 Product from 16 by 42 in. rolls. Gangue, altered schist.
 8 ft. by 22 in. pebble mill.
 Capacity, 180 tons per 24 hr.
 Charge, 10,000 lb. of pebbles.
 Speed, 27 rev. per minute.
 Horsepower, 36.
 Water, 60 to 65 per cent.
 Elevation of feed end, 1.5 in.
 Product desired to pass 30 mesh with a minimum of slime.

Mesh	Feed, Per Cent.	Discharge, Per Cent.
+ 4	0.4
+ 10	45.0
+ 20	32.7	2.2
+ 30	8.4	6.8
+ 40	1.7	7.0
+ 60	1.8	15.5
+ 80	0.8	8.9
+100	0.7	7.7
+150	0.8	8.5
+200	0.4	5.3
-200	6.3	38.1

Card 109. Federal Mining & Smelting Co., Wallace, Idaho.
 Middling from jigs. Gangue, quartzite and siderite.
 8 ft. by 22 in. pebble mill.
 Capacity, 110 to 115 tons per 24 hr.
 Charge, 10,000 lb. of pebbles.
 Speed, 28 rev. per minute.
 Horsepower, 35.8 net.
 Water, 60 per cent.
 Elevation of feed end, 0.
 Consumption of pebbles, 2 lb. per ton.
 Silex lining, life 13 months.
 Feed to mill through $\frac{3}{16}$ -in. mesh.

Mesh	Feed, Per Cent.	Discharge, Per Cent.
+ 10	41.2
+ 20	45.8	0.1
+ 30	8.0	1.3
+ 40	3.0	6.7
+ 60	1.3	15.6
+ 80	0.3	16.4
+100	14.5
+150	0.3	7.0
+200	13.4
-200	0.1	25.0

Card 75. Sept. 7 to 9, 1911. Federal Mining & Smelting Co., Wallace, Idaho.

Coarse Wilfley middling. Gangue, quartzite and siderite.

8 ft. by 22 in. pebble mill.

Capacity, 99.36 tons per 24 hr.

Charge, 5 tons of pebbles (approx.)

Speed, 28 rev. per minute.

Horsepower, 35.3 net.

Water, 55 per cent.

Elevation of feed end, 0.

Consumption of pebbles, 1.5 to 2 lb. per ton.

Feed to mill through 5-mm. mesh.

Mesh	Feed, Per Cent.	Discharge, Per Cent.
+ 20	35.0
+ 30	31.0	1.0
+ 40	18.5	3.0
+ 60	13.5	7.5
+ 80	1.0	7.0
+100	1.0	10.5
+200	4.0
-200	67.0

Card 136. Federal Mining & Smelting Co., Wallace, Idaho.

Jig middling. Gangue, quartzite.

8 ft. by 22 in. pebble mill.

Capacity, 111.5 tons per 24 hr.

Charge, 5 tons of pebbles.

Speed, 28 rev. per minute.

Horsepower, 35.3.

Water, 71.8 per cent.

Elevation of feed end, 2 in.

Consumption of pebbles, 2 lb. per ton.

Feed to mill through 4 mesh.

Mesh	Feed, Per Cent.	Discharge, Per Cent.
+ 10	30.1
+ 20	47.8
+ 30	12.6	1.0
+ 40	5.1	4.0
+ 60	2.5	12.0
+ 80	2.0	12.5
+100	13.0
+150	6.5
+200	12.0
-200	39.0

Card 150. Sept. 8, 1913. Vieille Montagne Zinc Co., Cumberland, England.

Zinc-lead ore. Gangue, siliceous limestone.

8 ft. by 22 in. pebble mill.

Capacity, 120 tons per 24 hr.

Charge, 3 tons of pebbles.

Speed, 29.5 rev. per minute.

Elevation of feed end, 4 in.

Feed to mill through $\frac{1}{2}$ -in. mesh.

Mesh	Feed, Per Cent.	Discharge, Per Cent.
+ 4	25.5
+ 6	30.0
+ 8	34.0
+ 10	7.5	0.2
+ 20	2.7	4.8
+ 40	18.0
+ 60	17.9
+ 80	15.2
+100	4.0
+200	25.8
-200	13.5

Card 34. Nov. 2, 1910. Calumet & Hecla Mining Co., Lake Linden, Mich.

Tailing from jigs. Gangue, Lake conglomerate.

8 ft. by 22 in. pebble mill.

Capacity 40 to 45 tons per 24 hr.

Charge, 3 tons of pebbles.

Speed, 27 rev. per minute.

Horsepower, 34 to 37.

Water, 40 per cent.

Elevation of feed end, 0.

Consumption of pebbles, 2 lb. per ton of ore.

Feed to mill through $\frac{1}{4}$ -in. mesh.

Mesh	Feed, Per Cent.	Discharge, Per Cent.
+ 6	0.25
+ 8	2.70
+ 10	5.10
+ 20	25.70	0.02
+ 40	36.40	0.15
+ 60	19.35	2.10
+ 80	7.00	6.55
+100	1.40	6.00
+150	1.45	16.90
+200	0.70	26.70
-200	0.20	40.80

TABLE I

No.	Diameter of Mill, Feet	Length Cylinder, Inches	Type	Mining Company	Gangue	Material	Cha Ball Pebbl Pou
122	4.5	13	Ball....	Vipond Porcupine Mines Co.	Quartz and basalt.....	Ore from mill bin.....	4,00
107	6	16	Ball....	Miami Copper Co.....	Siliceous porphyry.....	Ore from mill bin.....	8,00
155 ^a	6	16	Ball....	Britannia M. & S. Co..	Quartzose, very hard...	Jig tailing.....	8,20
192	6	16	Ball....	McIntyre Porcupine Mines.	Quartz and schist.....	Rock-crusher product...	8,00
156	6	16	Ball....	McIntyre Porcupine Mines.	Quartz and schist.....	Rock-crusher product...	8,00
191	6	16	Ball....	Buckhorn Mines Co....	Decomposed porphyry and basalt.	Rock-crusher product...	8,00
121	6	22	Pebble..	Bunker Hill & Sullivan.	Quartzite and siderite....	Middling from jigs and tables.	4,00
113	6	72	Pebble..	Vipond Porcupine Mines	Quartz and basalt.....	Oversize Colbath classi- fier.	9,00
108	8	22	Pebble..	Miami Copper Co.....	Altered schist.....	Product 16 by 42 in. rolls.	10,00
80	8	22	Pebble..	Miami Copper Co.....	Altered schist.....	Product 16 by 42 in. rolls.	10,00
109	8	22	Pebble..	Federal M. & S. Co.....	Quartzite and siderite....	Jig middling.....	10,00
75	8	22	Pebble..	Federal M. & S. Co.....	Quartzite and siderite....	Coarse Wilfley middling	10,00
136	8	22	Pebble..	Federal M. & S. Co.....	Quartzite and siderite...	Jig middling.....	10,00
150	8	22	Pebble..	Vieille Montagne Zinc Co.	Siliceous limestone.....	Jig middling.....	6,00
34	8	22	Pebble..	Calumet & Hecla.....	Conglomerate.....	Jig tailing.....	6,00
33	8	30	Pebble..	Copper Range Consol...	Amygdaloid.....	Jig tailing.....	10,00
142	8	36	Pebble..	Arizona Copper Co.....	Porphyry.....	Screened roll product...	10,50
135	7	144	Tube...	Federal M. & S. Co.....	Quartzite.....	Wilfley middling.....	18,00

^a See note on mesh card. Note R. M. E.

Card 135. Oct. 26, 1912. Federal Mining & Smelting Co., Morning, Idaho.

Wilfley middling. Gangue, quartzite.

7 ft. by 12 ft. tube mill.

Capacity, 124 tons per 24 hr.

Charge, 9 tons of pebbles.

Speed, 22.25 rev. per minute.

Horsepower, 86.

Water, 58.8 per cent.

Elevation of feed end, 0.

Consumption of pebbles, 4 to 5 lb. per ton of ore.

Mesh	Feed, Per Cent.	Discharge, Per Cent.
+ 10	1.5
+ 20	11.5
+ 30	23.5	0.5
+ 40	23.0	2.0
+ 60	23.5	7.0
+ 80	10.0	11.0
+100	4.5	15.0
+150	1.0	5.5
+200	1.0	14.5
-200	0.5	44.5

TABLE I—Continued

Mill.	Tons per 24 Hr.	Horsepower	Tons per Horsepower	Feed		Energy Units	Discharge		Per Cent. — 200 Mesh.	Energy Units	Difference, E. U.	Relative Mech. Effic. (R. M. E.)	Per Cent. Water in Feed	Elevation of Feed End, Inches	Pebbles or Balls Pounds per Ton
				All pass, Mm.	Average Size, Mm.		All Pass, Mm.	Average Size, Mm.							
	48	16	3.0	50.8	12.01	280.2	6.3	0.26	40.2 ^b	1,423.4	1,143.2	34.3	50
	351	35	10.0	38.1	6.01	665.3	12.7	0.86	21.8	1,247.0	581.7	58.3	50	2	0.578
	251	39	6.4	6.35	1.28	912.5	6.35	0.24	34.2 ^b	1,432.7	520.2	33.5	40	0	0.72
	150	36	4.2	50.8	12.7	313.1	6.35	0.19	41.07	1,705.1	1,392.0	58.0	50	1.25	0.5
	150	36	4.2	50.8	10.04	182.5	1.65	0.25	37.2	1,330.0	1,147.5	47.8	60	1.25	0.5
	160	33.25	4.8	38.1	7.09	525.4	3.2	0.16	28.0 ^c	1,535.6	1,010.2	48.6	80	1.5	0.45
	60.3	16(?)	5.2	3.0	0.58	1,092.8	0.36	0.09	41.8	1,686.9	594.1	30.9	75(?)	0.5
	40	30	1.3	3.0(?)	0.65	1,154.6	0.24	0.05	68.8	1,823.4	668.8	9.19	50	0.0
	101	36	2.8	25.4	1.36	973.8	1.65	0.19	35.9	1,573.2	599.4	16.8	63	1.5
	180	36	5.0	12.7	1.1	990.3	1.65	0.17	38.1	1,591.3	601.0	30.0	62.5	1.5
	112.5	35.8	3.1	4.7	1.12	943.2	1.65	0.14	25.0	1,594.7	651.5	20.5	60	0.0	2.0
	99.36	35.3	2.8	5.0	0.56	1,040.9	0.83	0.09	67.0	1,750.6	709.7	20.0	55	0.0	1.5-2
	111.5	35.3	3.3	4.7	0.99	986.4	0.83	0.12	39.0	1,649.4	663.0	21.7	71.8	2.0	2.0
.5	120	35(?)	3.4	12.7	3.15	632.5	2.4	0.21	13.5	1,492.3	859.8	29.5	4.0
	42.5	35.5	1.2	6.35	0.56	1,195.6	1.65	0.08	40.8	1,614.7	419.1	5.0	40	0.0	2.0
	65	46(?)	1.4	6.35	1.72	825.8	0.83	0.10	34.0	1,652.9	827.1	11.7	64	1.0	2.5
	208	55	3.8	12.7	2.89	786.5	2.36	0.27	34.7	1,502.1	715.6	27.1	59	0.0	2.4
.25	124	86	1.4	2.3	0.41	1,051.4	0.83	0.10	44.5	1,690.2	638.8	9.2	58.8	0.0	4-5

^b — 100 mesh. ^c — 150 mesh.

Table I is a summary of the data presented on the mesh cards. Table II is taken from Table I.

Apparently the ball mill does more work per unit of power input than the pebble mill, while the pebble mill is, in the same way, more efficient than the tube mill in the one instance cited. Nos. 107 and 142 show the effect of overloading the mill. The relative mechanical efficiencies in these two cases are raised above the average at the expense of the character of the product.

Card 33. Nov. 3, 1910. Copper Range Consolidated, Freda, Mich.

Tailing from jigs. Gangue, Lake amygdaloid.

8 ft. by 30 in. pebble mill.

Capacity, 65 tons per 24 hr.

Charge, 5 tons of pebbles.

Speed, 28 rev. per minute.

Water, 64 per cent.

Elevation of feed end, 1 in.

Consumption of pebbles, 2.5 lb. per ton of ore.

Feed to mill through $\frac{1}{4}$ -in. mesh.

Mesh	Feed, Per Cent.	Discharge, Per Cent.
+ 6	10.3
+ 8	30.0
+ 10	25.0
+ 20	29.0
+ 40	4.5	2.7
+ 60	0.5	8.5
+ 80	0.1	3.3
+100	9.4
+150	30.2
+200	10.9
-200	34.0

Card 142. Oct. 9, 1913. Arizona Copper Co., Morenci, Ariz.

Screened roll product. Gangue, porphyry.

8 ft. by 36 in. pebble mill.

Capacity, 208 tons per 24 hr.

Charge, 10,500 lb. of pebbles.

Speed, 29 rev. per minute.

Horsepower, 55.

Water, 59 per cent.

Elevation of feed end, 0.

Consumption of pebbles, 2.4 lb. per ton.

Feed to mill through $\frac{1}{2}$ -in. mesh.

Mesh	Feed, Per Cent.	Discharge, Per Cent.
+ $\frac{3}{8}$ in.	7.6
+ 4	26.0
+ 8	25.6
+ 10	12.1	0.6
+ 20	12.0	10.2
+ 30	4.8	13.9
+ 40	2.3	10.0
+ 60	2.0	11.7
+ 80	0.7	5.6
+100	0.4	4.4
+150	0.2	3.5
+200	0.5	5.4
-200	5.8	34.7

TABLE II

	6 ft. by 16 in. Ball Mill	8 ft. by 22 in. Pebble Mill
Average maximum size of feed, mm.....	44.5	9.7
Average size of feed, mm.....	9.0	1.26
Average maximum size of discharge, mm.....	6.0	1.5
Average size of product, mm.....	0.37	0.14
Average per cent. of -200 mesh in discharge.....	28.9 ^a	37.0
Average per cent. of -200 mesh in discharge, no slope.....		44.3
Average per cent. of -200 mesh in discharge, 0.5 to 4 in. slope.....		31.6
Reduction ratio, range.....	7 to 67	6 to 15
Reduction ratio, average.....	39.6	8
Average size of discharge, no slope, mm.....		0.10
Average size of discharge, slope 0.5 to 4 in.....		0.17
Average tonnage.....	203	110
Average tonnage at no slope.....		85
Average tonnage at 0.5 to 4 in. slope.....		128
Average horsepower.....	35.06	35.6
Average charge, balls or pebbles, tons.....	4	4.5
Average ball or pebble consumption, pounds per ton.....	0.51	1.94
Average relative mechanical efficiency.....	53.2	20.5
Average percentage of water in feed.....	60	58.7
Average revolutions per minute.....	28	27.8

^a Nos. 155 and 191 estimated.

Standardizing Rock-Crushing Tests

BY MYRON K. RODGERS,* B. S., LOS ANGELES, CAL.

(San Francisco Meeting, September, 1915)

IN rock- or ore-crushing tests all data, in order to be valuable for study and comparison, should be obtained and tabulated under conditions as uniform as possible. The results of many such tests have no value for comparison, because of incomplete data and the lack of uniform conditions, screens and scale of sizing, etc.

The purpose of this paper is to suggest a form of standard specifications which may be developed by the Milling Committee and approved by the A. I. M. E. for rock-crushing tests. It is not intended to discuss any theories of rock crushing.

Reports of rock-crushing tests should include the following details:

1. Description of the machine employed (jaw or gyratory crushers, rolls, stamps, tube mills, Chilean mills, etc.).
2. Method and material (timber, concrete, etc.) of foundation. Much power is dissipated in the vibration of poor foundations.
3. Locality from which the rock or ore was obtained, and geological, mineralogical, and physical characters of the material.
4. The power consumption of the machine running with no load and with full load, the unit of power being 1 hp. per 24 hr.
5. The capacity of the machine in tons (of 2000 lb.) per 24 hr.
6. The duty in tons per horsepower-day.
7. The screen analysis of feed and product by the proposed A. I. M. E. standard screen scale.

On the foregoing statement, the following remarks are offered.

Toughness of Rock

Table I shows the relative crushing duty of the same machine on three different characters of rock, all other crushing conditions being the same.

This table indicates that the specific resistance of the rock to crushing, may affect the crushing duty of a unit of power as much as 300 per cent. Perhaps this quality may be called toughness. It is not a function of hardness only. For comparison of tests, rocks should be divided into

*Mining Engineer.

TABLE I.—*Relative Crushing Duty of the same Rock-Crushing Machine on Three Different Characters of Rock, All Other Crushing Conditions Being the Same*

Rock and Feed	Mill	Per Cent. Solids	Hp. of Machine	Tons per Machine, 24 hr.	Tons per Hp.-Day	Tons 200 Mesh per Hp.-Day	Screen Scale + 4 + 10 + 20 + 40 + 60 + 100 + 200 - 200									
Calumet & Hecla conglomerate, ¼ mesh.	Conical tube 8 ft. by 30 in.	40	50.0	65	1.30	0.44	Feed									
							0	65	29	5	1
Cœur d'Alène quartzite, F. M. & S. Co., 5 mm.	Conical tube 8 ft by 22 in. 28 r.p.m.	43	47.5	112	2.50	0.67	Feed									
							0	43	43	11	1	1	1	1	1	1
Miami, ¼ mesh.	Conical tube 8 ft. by 22 in.	27	47.5	229	4.85	1.80	Feed									
							4	45	33	10	2	2	1	7	7	7
							Product									
							2	22	18	19	12	27	27	27
							2	14	12	17	14	38	38	38

Relative crushing duty of 1 hp. due to character of ore only:

Calumet & Hecla conglomerate	1.00
Cœur d'Alène quartzite	1.92
Miami.....	3.75

at least three classes in this respect; and the class to which a particular rock belongs might be determined by its crushing strength per square inch.

Screens

For screen analysis, a standard screen scale should be used, having a uniform ratio between the different sizes.

The results of an investigation made by the U. S. Bureau of Standards, with several of the large mining companies, where 240- to 260-mesh screens are used in connection with flotation tests, and including several cement manufacturers and screen manufacturers, are that a screen analysis for all rock-crushing tests, and practically all testing of materials, will be covered by a screen scale based on the U. S. Bureau of Standards, 200-mesh screen with a square opening of 0.0029 in. and a wire of 0.0021 in. diameter, advancing with a ratio of 1.414 (*i.e.*, the square root of 2, suggested by Rittinger) between the successive sieve openings to an opening 4.20 in. square. Where finer than 0.0029 in. opening (200 mesh) is required, 0.0020 in. opening (280 mesh) can be used, retaining the above ratio.

TABLE II.—*Standard Screen Scale*

Mesh per Linear Inch	Opening, Inches	Opening, Millimeters	Diameter of Wire, Inches	Diameter of Wire, Millimeters	Area of Openings, Square Inches
	4.20	106.60	0.375	9.52	17.64
	2.97	75.39	0.207	5.26	8.82
	2.10	53.33	0.192	4.88	4.41
	1.49	37.73	0.149	3.78	2.20
	1.05	26.67	0.149	3.78	1.10
	0.742	18.85	0.135	3.43	0.551
	0.525	13.33	0.105	2.67	0.276
	0.371	9.423	0.092	2.34	0.138
3	0.263	6.680	0.070	1.78	0.069
4	0.185	4.699	0.065	1.65	0.034
6	0.131	3.327	0.036	0.91	0.017
8	0.093	2.362	0.032	0.81	0.0086
10	0.065	1.651	0.035	0.89	0.0042
14	0.046	1.168	0.025	0.64	0.0021
20	0.0328	0.833	0.0172	0.44	0.00108
28	0.0232	0.589	0.0125	0.32	0.00054
35	0.0164	0.417	0.0122	0.31	0.00027
48	0.0116	0.295	0.0092	0.23	0.000135
65	0.0082	0.208	0.0072	0.18	0.0000672
100	0.0058	0.147	0.0042	0.11	0.0000336
150	0.0041	0.104	0.0026	0.07	0.0000168
200	0.0029	0.074	0.0021	0.05	0.0000084

If the analysis is to be carried finer than 0.0029 in. (200 mesh), the next finer sieve-opening in the Screen Scale Series, is 0.002 in. (280 mesh).

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Conveyor-Belt Calculating Chart

BY J. D. MOONEY AND D. L. DARNELL,* AKRON, OHIO

(San Francisco Meeting, September, 1915)

THE accompanying chart has been drawn for the convenience of engineers as a means of quickly determining the correct number of plies of conveyor belts operating under specific conditions.

The calculations are based on the average safe strength (factor of safety, 15) of the various standard rubber conveyor belts.

The calculations assume maximum loading conditions; that is, the belt is considered as carrying the greatest load that it will handle without spillage at ordinary belt speeds. This not only produces the most economical operating conditions, but also the maximum tension in the belt.

The chart is a graphical representation of the formula:

Where, $p = kgW(L + 10H)$

p = the correct number of plies

k = a constant, depending on the type of drive

g = the weight in pounds per cubic foot of material handled

W = the width of the belt in inches

L = the length of the belt in feet (approximately twice the center distance).

H = the difference in elevation between the head and tail pulleys, in feet.

For a simple drive, with a bare pulley, $k = \frac{1}{250,000}$

For a simple drive, with a rubber-lagged pulley, $k = \frac{1}{300,000}$

For a tandem drive, with bare pulleys, $k = \frac{1}{375,000}$

For a tandem drive, with rubber-lagged pulleys, $k = \frac{1}{455,000}$

The chart is drawn for a simple drive, with a bare pulley ($k = \frac{1}{250,000}$);

therefore, the number of plies obtained from the chart should be multiplied by the factor 0.83 or $\frac{5}{6}$ for simple, lagged drive; the factor 0.67 or $\frac{2}{3}$ for tandem, bare; and the factor 0.55 or $1\frac{1}{20}$, for tandem, lagged.

The formula $p = kgW(L + 10H)$ has been developed mathematically

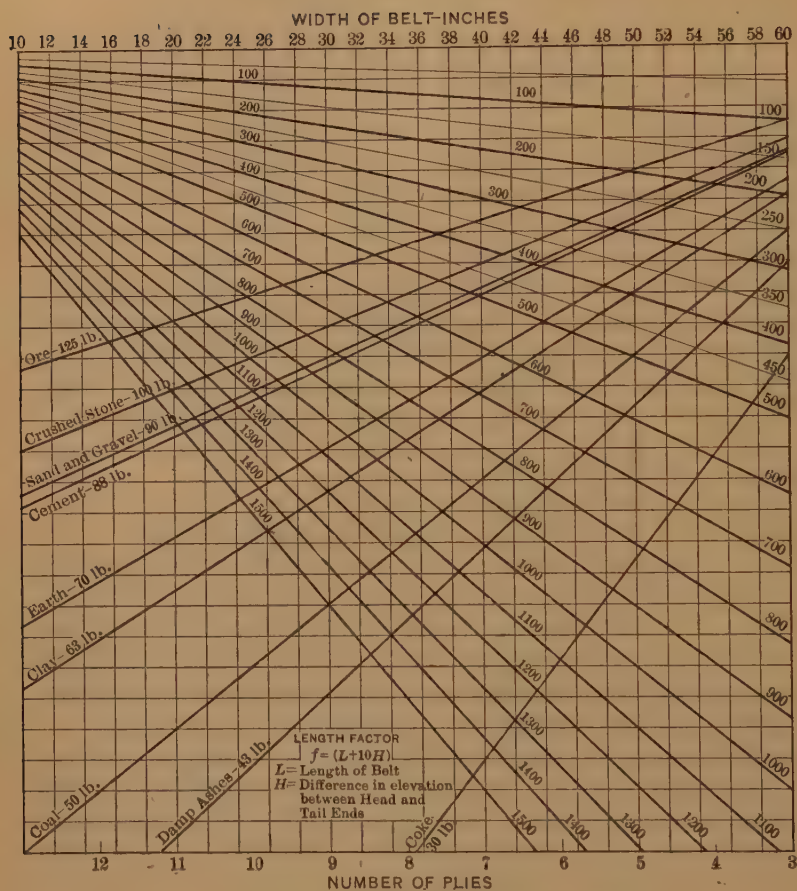


FIG. 1.—CONVEYOR BELT CALCULATING CHART.

from the following formulas, which have been found to work very satisfactorily in average good practice:

$$U = \frac{0.08W^2Sg}{5000}$$

$$HP = \left(\frac{0.02l}{100} + \frac{0.01H}{10} \right) U$$

$$T = \frac{C \times HP \times 100}{S}$$

$$p = \frac{T}{24W}$$

Where, U = Capacity in tons per hour
 W = Width of belt in inches
 S = Belt speed in feet per minute
 g = Weight per cubic foot of material handled
 HP = Horsepower developed in driving conveyor belt
 l = Length of the conveyor, in feet (approximately $\frac{1}{2}L$)
 H = The difference in elevation between the head and tail pulleys, in feet
 T = The total tension in the belt, in pounds
 p = Correct number of plies
 C = The constant of the drive.

For a simple drive, bare pulley,	$C = 600$
For a simple drive, rubber-lagged pulley,	$C = 500$
For a tandem drive, bare pulleys,	$C = 400$
For a tandem drive, rubber-lagged pulleys,	$C = 330$

The length factor, $f = (L + 10H)$, represented on the chart by the lines 500, 600, 700, etc., is a developed factor equal to the sum of the length of the belt and ten times the difference in elevation between the head and tail pulleys.

To find the correct number of plies for a conveyor belt, knowing the width, the length, the difference in elevation between the head and tail ends, and the kind of material to be handled:

Start from the width given at the top of the chart and move down until this line intersects the line corresponding to the proper length factor; then move either right or left until the line corresponding to the given material is met; then move down again to the scale of plies, where the next largest figure will give the correct number of plies.

For example: To find the correct number of plies for a conveyor belt 36 in. wide and 300 ft. long, with 20 ft. difference in elevation; handling sand and gravel.

Follow the line from the 36-in. width downward until it intersects the 500 length factor line; then follow to the right until the sand and gravel line is intersected; then down to the ply scale, where the ply will be found to be 7.

The British Columbia Copper Co.'s Smelter, Greenwood, B. C.

BY FREDERIC K. BRUNTON,* B.S., VICTOR, COLO.

I. INTRODUCTION

THE smelting plant of the British Columbia Copper Co. at Greenwood, B. C., now closed because of the decline in the price of copper due to the European war, is of special interest to metallurgists for several reasons.

It was successfully smelting in blast furnaces the lowest grade copper ore of all plants in America. In order to do so, it had to run at very high efficiency, which necessarily required a large tonnage per square foot of hearth area, together with the minimum amount of labor and other costs.

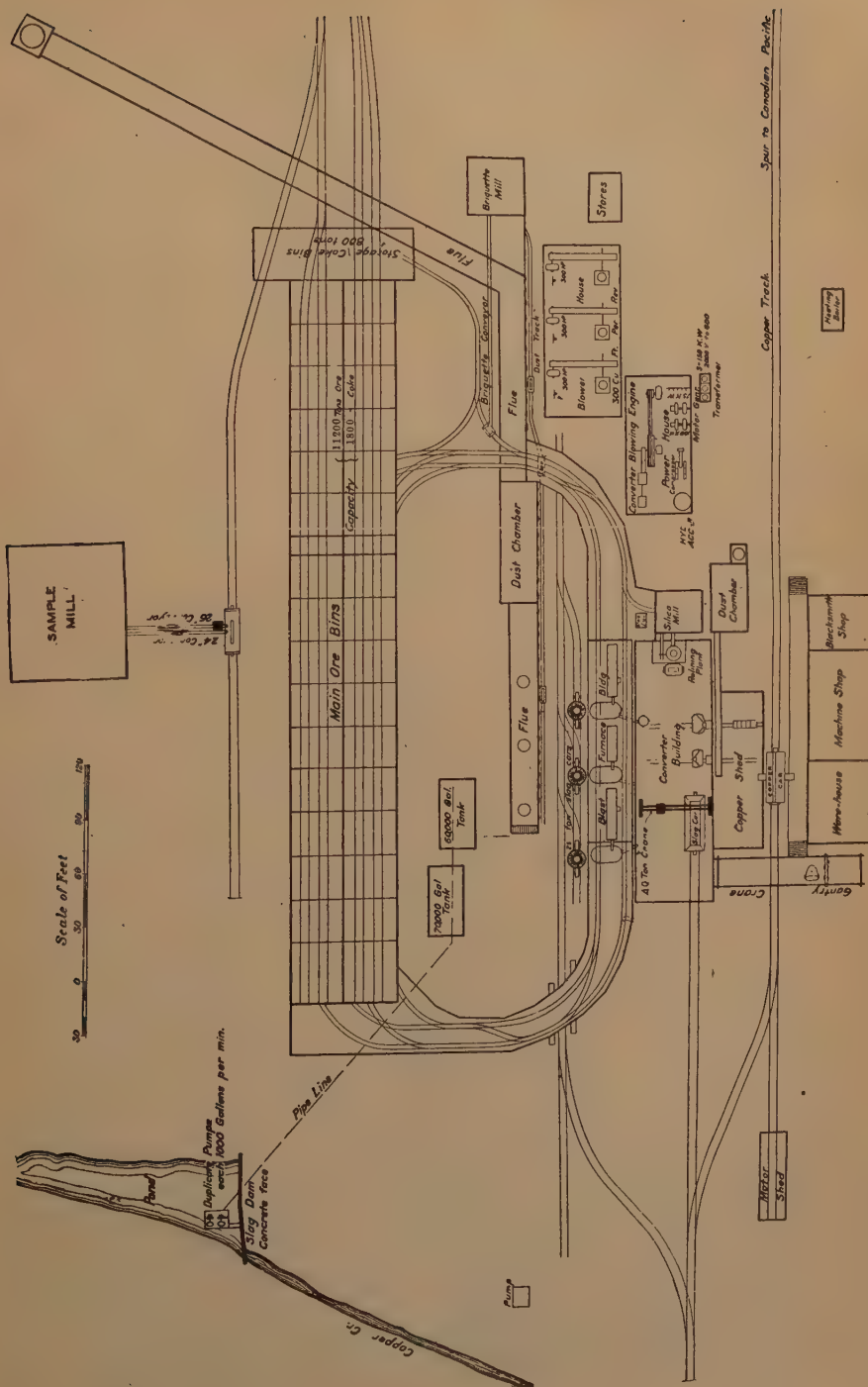
The furnaces smelted daily 2,250 tons of ore (6.62 tons per sq. ft. of hearth area), carrying 0.85 per cent. of copper, at a smelting cost of \$1.18 per ton. The entire plant required 130 men to operate it and keep up repairs, showing a labor efficiency of about 17.5 tons per man per day.

In the present paper, the method of obtaining these results is shown. Most of the information contained in this description I obtained as Assistant Superintendent, during the two years preceding the present shutdown. Other sources of information to which I am indebted are: The British Columbia Copper Co., Ltd., by Alfred W. G. Wilson, in *The Copper Smelting Industry of Canada*; Greenwood Smelting Works, by J. E. McAllister, *Engineering and Mining Journal*, vol. xci, pp. 1011 to 1015 (May 20, 1911); and Description of the Copper Smelter of The British Columbia Copper Co., by W. L. Bell, in *Transactions of the Canadian Mining Institute*, vol. xvi, pp. 151 to 154 (1913).

II. LOCATION AND CONNECTIONS

The smelter of this company is about half a mile south of Greenwood, in the Boundary district of British Columbia, on the Columbia & Western branch of the Canadian Pacific Railway. It was originally built in 1900 by Paul Johnson, to handle 600 tons of ore a day in two small furnaces of 300 tons capacity; but in 1907 the old furnaces were torn out, the plant was remodeled, and three blast furnaces, 48 by 240 in. at the tuyères, were installed. In 1910, two of these were enlarged to 51 by

* Metallurgical Engineer, Copeland Ore Sampling Co.



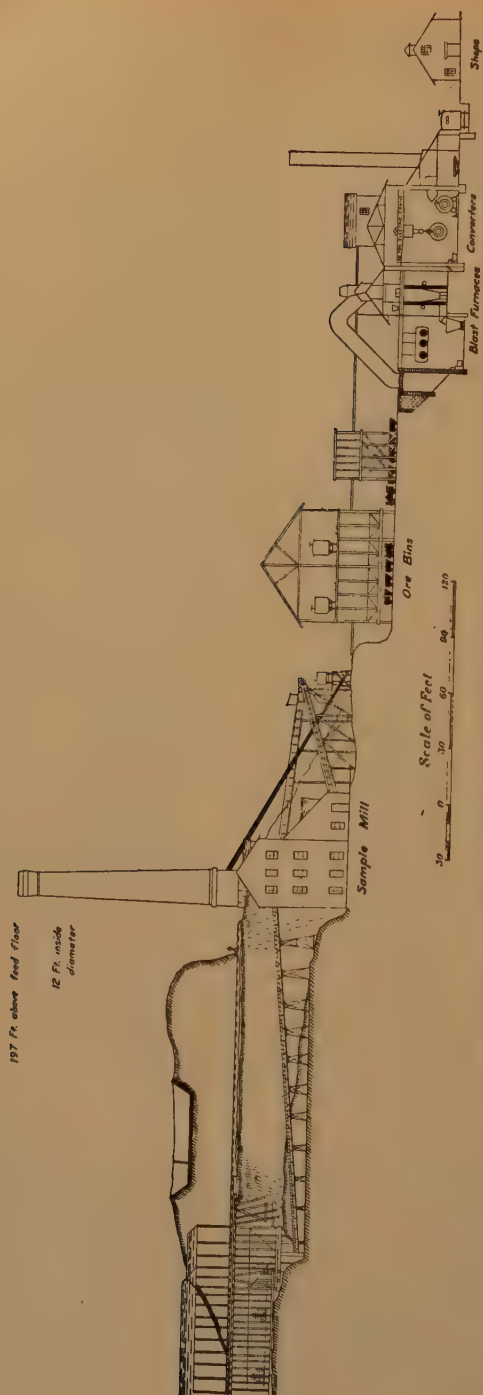


FIG. 2.—SECTION THROUGH REDUCTION WORKS AT GREENWOOD.

360 in. and the other to 51 by 240 in., making the total smelting capacity about 2,400 tons per day.

The Canadian Pacific Railroad delivered ore and coke to the smelter, which is situated on a hillside above the valley, by eight of the nine spurs of track which enter the plant at three different levels; the ninth spur, coming into the plant on the lowest level, took care of the incoming supplies, converter slag, and copper shipments. The ore as received

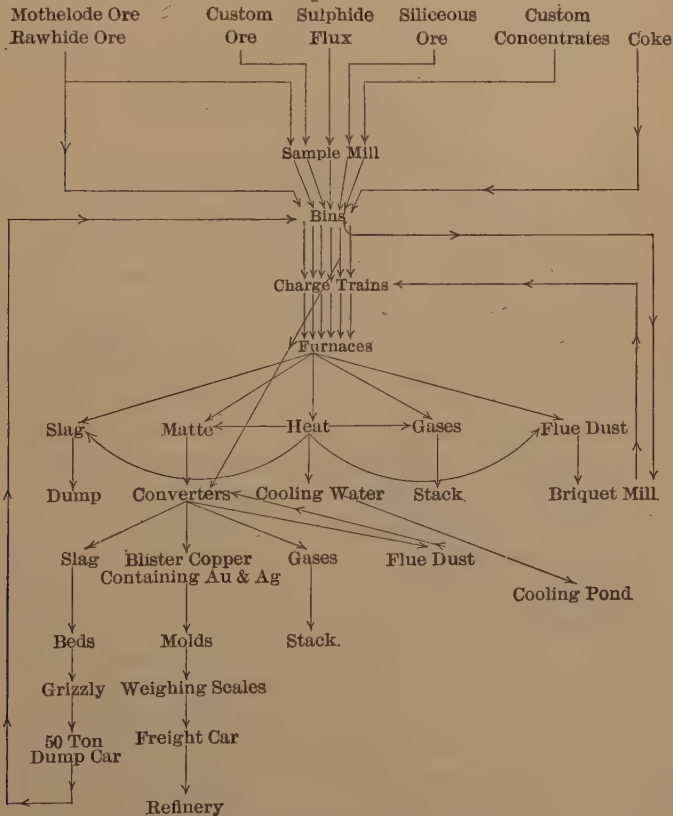


FIG. 3.—GENERAL FLOW SHEET OF THE BRITISH COLUMBIA COPPER CO.'S SMELTER, GREENWOOD, B. C.

was either dumped into the bins above the sampling works or directly into stock bins above the charging floor of the smelter. The sampling mill was served by three of the nine spurs, one of which, running over a 200-ton bin, brought ores from the company's own mines, while another, running over two small bins of about 60 tons capacity, was used for custom ores.

On the intermediate level, 48 ft. below the tracks serving the sampling mill, are five spurs, four of which run over the main stock bins and one

under the discharge belt of the sampling mill, so that rejects from the mill can be switched to the bins very handily.

The main spur coming into the plant at this level is equipped with a standard 100-ton self-registering track scale, so that all material can be weighed if desired.

Fig. 1 is a plan and Fig. 2 a section of the plant. The general flow sheet of the smelter is shown in Fig. 3 and the flow sheet of the sampling mill in Fig. 4.

III. SAMPLING, STORAGE, AND CHARGING

1. *Sampling Mill*

The building is 65 by 79 ft. in area, and three stories high. The ore from the sample bins, whether custom or company ore, was discharged through chutes into a 36 by 24 in. Farrell-Bacon crusher, crushing to 8 in., driven from a countershaft by a 100-hp. Allis-Chalmers Bullock, variable speed, 2,200-volt, induction motor. The crushed ore went to the sample mill on a 30-in. belt conveyor, 225 ft. between centers, driven from the mill line shaft. If it was company ore, this conveyor delivered it into a bucket sampler which cut out 11.5 per cent. for a sample, while the reject, falling on a 26-in. belt conveyor, 116 ft. between centers, driven by a 15-hp., 550-volt, motor, was delivered to a standard 50-ton steel dump car which, when full, was switched to the smelter storage bins. If the ore from the jaw crusher was custom ore, it passed by the bucket sampler and was delivered through a chute into a No. 5 Gates crusher, crushing to 4 in. The 11½ per cent. sample of company ore was delivered to the same crusher by the bucket sampler; and either ore, after leaving the crusher, went through a double-spouted Johnson 48-in. sampler, taking a 10 or a 20 per cent. cut, as desired, the reject going to a bin which discharged on to a 24-in. belt conveyor 120 ft. long, driven by a 10-hp., 550-volt, induction motor, delivering to the dump car mentioned above.

The sample from the 48-in. Johnson sampler was passed through a No. 2 Gates crusher, crushing to 2 in., and then a 40-in. Johnson sampler taking a 20 per cent. cut. This and all other rejects were delivered to the 24-in. belt conveyor above referred to, while the sample was elevated by a bucket elevator to a No. 0 Gates crusher, crushing to 1 in., and then passed through a set of 10 by 18 in. rolls, crushing to 0.5 in., after which it was delivered to a 24-in. three-spouted Johnson sampler taking a 20 per cent. sample. The reject went to the dump car and the sample to a sample car in the sample room, from which it was taken to a steel floor and cut down with a Case riffle to about 30 lb., which was then taken to the bucking room, where the sample was alternately crushed and riffled until the pulp was of suitable size for the assay office.

The fineness of crushing between each two riffings depended upon the grade of the ore. The final pulp was passed for company ore through an 80-mesh, and for custom ore through a 150-mesh sieve.

The entire mill was operated from a line shaft driven by a 100-hp., 550-volt, induction motor, the low voltage being used on account of the amount of dust in the air. The sampling mill had a capacity of 1,000

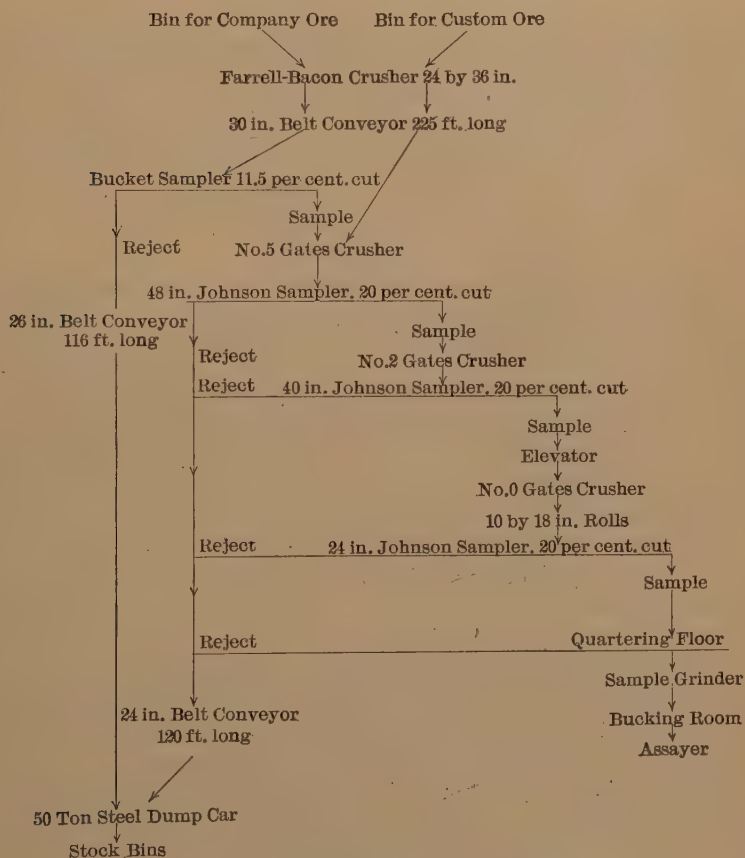


FIG. 4.—SAMPLING MILL FLOW SHEET.

tons per 8-hr. shift, and required three men besides the sample bucker. The Johnson samplers referred to were two and three spout vertical radial cutting machines of the well-known Vezin type. The bucket sampler consisted of two steel boxes 2 ft. 6 in. long by 2 ft. wide by 10 in. deep, attached to double sprocket chains and moving around the sides of an inverted isosceles triangle, cutting at regular intervals the entire stream of ore delivered by the conveyor belt.

2. Bins

The 32 bins are built in two lines of 16 each, parallel to and touching each other, and served by four parallel standard-gage spurs, these being four of the five entering the smeltery on the intermediate level. The back or south end of this line of bins consists of 15 bins of 600 tons capacity each, 6 of 300 tons, and 4 of 100 tons, used for ore, while at the front or north end are 10 bins for coke, each of which will hold 180 tons, and one large box bin for clay, making the total capacity of the bins 11,200 tons of ore and 1,800 tons of coke. Coke for stock purposes was dumped on the ground from the fifth railway spur, which also served the sampling mill, and about 6,000 tons of coke could be stored there in an emergency. The ore and coke were all dumped by a crew of four men and a foreman, who also looked after the sampling-mill crew.

3. Charging Equipment

The ore and coke were drawn from the bins into charging cars running on four parallel, 36-in. gage, electric tramway lines on the feed-floor level, 31 ft. above the top of the ore bins. The bins and furnaces being on opposite sides of an oval, the tracks are so arranged that, with switches, a train may run on any track under the bins and come in to feed on either side of any furnace, the trains running completely around the oval. There was a charging train for each furnace, consisting of four ore and four coke cars for each of the 30-ft. furnaces and three ore and three coke cars for the 20-ft. furnace, each train being, roughly, twice the length of the furnace it fed. Each track is equipped with three sets of automatic platform scales, for weighing the ore and coke, placed on a tangent under the bins.

The cars, approximately V-shaped in cross-section, are side-dumping, being released by a foot trip in the bottom. They have a capacity of 50 cu. ft. of material and, when made up into a train, were hauled by a 7.5-ton electric locomotive, handled by a train crew of three men, a motorman, a head loader, and a second loader.

4. Charging

The crew usually made up the charge by loading the coke in the last half of the train and weighing it; then running ahead and drawing the ore into the front cars and weighing it on the nearest platform scale; and then running around the oval to the furnaces, where, with the help of the feeder, the coke cars were dumped first; the train was then backed up and dumped the ore, after which it proceeded around to the bins for another charge. The charge usually consisted of 5,000 lb. of ore per car and about 13 per cent., or 650 lb., of coke. The loaders were very

expert at estimating their charge as they drew it from the bins and usually ran in very nearly the amount required, so that when it was weighed they only had to take off or add from 5 to 10 lb.

IV. BLAST-FURNACE DEPARTMENT

1. Blast Furnaces

The blast-furnace building is 150 ft. long by 60 ft. wide and contains three water-jacketed blast furnaces placed end to end, with space between them for the minor axis of a 10 by 18 ft. oval settler. The two outside furnaces, Nos. 1 and 3, are each 51 by 360 in., while the middle one, or No. 2, is 51 by 240 in. in area at the tuyères. The vertical distance from the center of tuyères to the feed floor is 16 ft., and to the sole plate 37 in., the other furnace dimensions being as follows:

	30-ft. Furnace	20-ft. Furnace
Hearth area.....	127.5 sq. ft.	85 sq. ft.
Center tuyères to tapping floor.....	5 ft. 3 in.	5 ft. 3 in.
Height bottom jackets.....	9 ft. 0 in.	9 ft. 3 in.
Width of side jackets.....	3 ft. 4 in.	3 ft. 4 in.
Width of end jackets, bottom.....	3 ft. 8 in.	3 ft. 8 in.
Width of end jackets, bottom.....	6 ft. 2 in.	6 ft. 2 in.
Number of tuyères.....	72	48
Diameter of tuyères.....	4 in. bushd to $3\frac{1}{4}$ in.	4 in.
Area of tuyères.....	597.4 sq. in.	602.9 sq. in.
Tuyère area per sq. ft. of hearth area.....	4.65 sq. in.	7.09 sq. in.
Center line to center line tuyères.....	9.25 in.	9.25 in.
Water space in jacket, 4 in.; plate used on inside $\frac{3}{8}$ in.; on outside, $\frac{3}{8}$ in.		

The furnaces and settlers are set upon concrete foundations about 4 ft. above the converter floor. The settlers are 10 ft. wide, 18 ft. long, and 4 ft. 8 in. deep, lined on the bottom with a layer 6 in. deep of chrome bats ground up and tamped in with clay, and this covered with a course of firebrick set on end, and on the sides with two courses of chrome brick, backed by crushed silica, for about 22 in. above the bottom. Above that line silica or firebrick are used instead of the chrome brick, except directly under the slag and matte spout of the furnace, and under the settler slag-discharge spout. The top of each furnace is connected with a dust chamber of 550 sq. ft. cross-section by a sheet-steel downtake, 7 ft. in diameter, in the form of an inverted V. This dust chamber is hopper bottomed, and the dust was drawn off in a small steel mine car and trammed to the briquet mill, where it was mixed with about 3 per cent. water and made into briquets in a White three-plunger press, having a capacity of 5 tons per hour, and driven by a 40-hp., 550-volt, induction motor. The briquets were fed back to

the furnaces again as extra coke when needed, since they contained about 70 per cent. of coke dust. The flue dust recovered amounted to about $1\frac{1}{2}$ per cent. of the total charge fed to the furnaces. The gases from the dust chamber passed through an inclined flue 180 sq. ft. in cross-section to the stack, which is 121 ft. high, with an inside diameter of 12 ft. The total height of stack above the top of the furnace-charge floor is 200 ft., which, under normal conditions, gave a good draft and kept the feed floor free from fumes.

2. Operation of the Furnaces

The ore and coke were charged into the furnaces from the side, the furnace door being lifted by compressed air. The cars of coke were dumped first, and the train was then backed up and the ore dumped. The charge fell against baffle plates, suspended on both sides of the furnace on 4-in. heavy hydraulic pipe which is supported by the end castings of the furnace on a level with the charge floor. To insure further the even distribution of the feed, the furnaces were fed first on one side, then on the other. When the plant was running under normal conditions, the two 30-ft. furnaces smelted from 12 to 16 5,000-lb. charges an hour, or about 112 charges per shift, each, and the 20-footer from 8 to 12 5,000-lb. charges per hour, or 80 per 8-hr. shift, making a total of 912 charges of 5,000 lb., or 2,250 tons, per day. The blast varied from 16 to 24 oz., according to conditions, and was supplied by three No. 10 Roots blowers, each having a capacity of 25,000 cu. ft. per minute running at 88 rev. per minute and belted by a 40-in. leather belt to a 300-hp., 2,200-volt, induction motor. The blowers are so arranged that, through a common header, any blower could be used on any furnace, the air going in iron pipes, 43 in. in diameter, from the header to the furnace bustle pipe, which extended around both sides of the furnace.

The slag and matte ran out of the furnaces at the end through a water-jacketed, copper breast jacket and thence through a water-jacketed spout with a water-cooled copper lip, into the settler. The spouts are set to have about a 7-in. trap. The slag overflowed continuously into a cast-iron pot of 250 cu. ft. capacity, holding from 20 to 25 tons of slag, and mounted on standard-gage trucks, in which it was hauled to the dump by a 35-ton, 240 hp., Baldwin-Westinghouse locomotive, and there dumped by means of a 15-hp., D. C. motor operated from the cab of the locomotive through "jumpers." While one pot was being removed from the settler and another one spotted, the slag was caught in a cast-iron spoon, supported upon a swinging arm and operated by hand from the side of each settler, which intercepted and received the slag stream and dumped its contents into the empty pot after the change was completed.

3. Furnace Charge

An average furnace charge was about as follows:

Ore	Lb.	SiO ₂		Fe		CaO		S		Cu	
		Per Cent.	Lb.	Per Cent.	Lb.	Per Cent.	Lb.	Per Cent.	Lb.	Per Cent.	Lb.
Motherlode.....	2,500	32	800	18	450	22	550	2	50	1.0	25.0
Rawhide.....	1,600	36	576	16	256	19	304	3	48	1.2	19.2
Napoleon.....	500	30	150	30	150	7	35	17	85	0.2	0.5
Republic, SiO ₂	400	80	320	7	28
Total.....	5,000		1,846		856		917		183		44.7

This gave approximately 4,300 lb. of slag carrying off 0.23 per cent. or about 10 lb. Cu, and leaving 34.7 lb. for the matte, which required one-fourth of its weight, or 8.7 lb., of sulphur. By reason of the poor construction of the furnaces for handling ores so low in sulphur, 88 per cent., or 160 lb., of the sulphur was volatilized in the furnace, leaving 23 lb. for the matte. Since the Cu took 8.7 lb. of S, there was left for iron 14.3 lb. of S, requiring 1.75 times its weight, or 25.0 of iron. The matte, therefore, contained: Cu, 34.7; S, 23.0; and Fe, 25.0 lb., and assayed 42 per cent. Cu. The slag ran approximately as follows: SiO₂, 42.7; Fe, 19.7; CaO, 21.2; Al₂O₃, 9.0; and Cu, 0.23 per cent.

In the above charge the most variable factor was the Motherlode ore, in which SiO₂ varied from 17 to 38, iron from 15 to 33, CaO from 7 to 22 per cent. Since this ore would change, sometimes in an hour, from a lime gangue to an iron gangue carrying about 30 per cent. magnetite, the furnaces required constant watching. The storage facilities were so small that the ore was always smelted before the assays were out; and this made it necessary to be able to determine the composition of the charge by the physical appearance of the matte and slag and the condition of the furnaces so that the charge could be changed to meet requirements at a moment's notice.

The labor required to operate the three furnaces per shift was as follows:

Kind of Labor	Number of Men	Wages per Shift	Total Wages per Shift
Shift bosses.....	1	\$5.25	\$5.25
Furnace men.....	3	4.00	12.00
Furnace helpers.....	3	3.00	9.00
Slag motorman.....	1	3.40	3.40
Slag switchman.....	1	3.00	3.00
Charge motormen.....	3	3.15	9.45
Head loaders.....	3	3.15	9.45
Second loaders.....	3	3.00	9.00
Feeders.....	1	4.00	4.00
Binman.....	1	2.75	2.75
Power house.....	1	3.40	3.40
Total.....	21		\$70.70

In addition, there were, on the day shift only, four men in the briquet mill and four men grading on the slag track. Of these, six received \$2.75 and the other two \$3.50 a day, making a total of \$23.50. These two crews were also used, whenever needed, in general roustabout work around the plant; so that not all of their time was really chargeable to the furnaces.

4. *Résumé of Furnace Operating Data*

Tons smelted per day.....	2,250.
Tons smelted per square foot of hearth area, average...	6.62
Tons smelted per square foot of hearth area, maximum.	8.70
Tons smelted per man per day.....	35.70
Cu on charge, per cent.....	0.8 to 1.2
Cu in matte, per cent.....	30.0 to 45.0
Cu in slag, per cent.....	0.22 to 0.27
S on charge, per cent.....	2.00
S burnt off, per cent.....	85.00 to 90.00
Coke used on charge, per cent.....	12.00 to 14.00
Coke ash, per cent.....	20.00 to 28.00
Blast, cubic feet per minute.....	25,000
Blast, temperature.....	Atmospheric
Cooling water for jackets, gallons per minute.....	2,500
Men per 8-hr. shift.....	21
Matte, per cent. of total charge.....	1.65
Matte, specific gravity.....	5.0
Slag, per cent. SiO_2	38 to 45
Slag, per cent. Fe.....	13 to 20
Slag, per cent. CaO	20 to 26
Slag, per cent. Al_2O_3	6 to 9
Slag, specific gravity.....	3.2

V. CONVERTER DEPARTMENT

The converter building is 150 ft. long by 44 ft. in width, stands parallel to and about 4 ft. lower than the blast-furnace tapping floor, and contains the following equipment:

Two hydraulic converter stands; seven converter shells, each 84 by 126 in.; a 40-ton Niles electric traveling crane, a 6-ft. Carlin silica mill driven by a 40-hp., 550-volt, induction motor, a pneumatic tamping machine, copper casting trucks, slag and matte pots, and a set of platform scales for weighing the blister copper. The converters were lined with siliceous gold ores from Republic, Wash., or from the Snowstorm mine in Idaho, mixed with local clay, as a binder, in the Carlin mill. The life of a lining was from two to three charges.

The matte, containing from 30 to 45 per cent. Cu, was tapped from the settlers into 5-ton ladles, handled by the Niles crane, and poured into one of the converters, where from two to three ladlefuls were blown to blister copper at one heat. The slag was poured off into old matte

ladles and was cast in hollows in the ground, into buttons, which when cool enough, were lifted by the crane, placed on grizzlies over 50-ton steel dump cars, and there broken up, with hammers, small enough to pass through the grizzlies. These cars, when full, were switched out over the ninth and lowest railway spur, which is 61 ft. below the intermediate level referred to before, and taken up to the main smelter bins, where the material was dumped and subsequently smelted in the furnaces.

The blister copper was cast into slabs, weighing approximately 360 lb. each, in cast-iron molds mounted on trucks (five molds to each truck) which ran on tracks under the converters. The copper slabs were weighed; loaded in box cars spotted on the lower spur, near the copper scales; and shipped East to be refined. This lower spur runs between the converter building and the machine shops and warehouse, and was also used for receiving incoming supplies.

The gases from the converters passed off into an iron flue, connected with an expansion chamber of 281 ft. cross-section, and thence into a steel stack 72 in. in diameter and 75 ft. high; the flue dust collected being fed back into the molten converter charge.

The converter department produced per day about 30,000 lb. of blister copper, carrying about 7 oz. of gold and 30 of silver per ton. It required a crew of 21 men which, divided as follows into two 8-hr. shifts, was able to handle all the matte produced:

Kind of Labor	Day Shift,	Afternoon Shift
	7 A.M. to 3 P.M.	3 P.M. to 11 P.M.
Foremen.....	1	0
Converters.....	2	2
Crane.....	2	2
Laborers.....	3	1
Lining.....	5	3
Total.....	13	8

VI. POWER

Power was purchased from the West Kootenay Light & Power Co. on a wattmeter basis and usually came from the Bonnington Falls power plant, about 85 miles away. It was brought by two 3-phase, 60-cycle, 60,000-volt high-tension lines, to their Greenwood substation, where it was transformed to 2,200 volts, at which intensity it was delivered to the smelter power house, to be used direct, or further transformed to suit the individual motor supplied.

The power house is divided into two parts, one of which contains the Roots blowers for the furnaces, already described, and the other a cross-compound Nordberg blowing engine for the converters, having a 40-in. air cylinder, 42-in. stroke, and a capacity of 5,000 cu. ft. per minute at from 8 to 10 lb. pressure per square inch, driven with a rope drive by a 300-hp., 2,200-volt, Canadian General Electric variable speed

induction motor. A compound duplex Canadian Rand compressor of 340 cu. ft. capacity, belt-driven by a 50-hp., 550-volt, induction motor running at 150 rev. per minute, supplied air at 80 lb. pressure for the converter tampers, the blast-furnace doors, the pneumatic hammer in the blacksmith shop, and for odd jobs around the plant where compressed air was required. Direct current at 250 volts for the crane, slag locomotive, charge-train motors, and an electric arc for burning frozen tap holes and furnace connections, was supplied by two of three motor-generator sets working in parallel, one of 75 kw., and the other of 150 kw. capacity, the third, of 100 kw. capacity, being held in reserve. The Gould hydraulic accumulator, with a ram 24 in. in diameter and a 10-ft. stroke, for tilting the converters, was operated by a triplex plunger pump, 5 in. in diameter, with an 8-in. stroke, running at 60 rev. per minute, with a capacity of 120 gal. per minute at 200 lb. pressure, belt-driven by a 20-hp., 550-volt, induction motor.

VII. GENERAL MECHANICAL EQUIPMENT

The plant has a well-equipped machine and blacksmith shop, where almost all repairs could be made. The shop contains one large and one small Bertram lathe, one large and one small radial-arm drill press, one large and one small pipe threader, and a planer, a bolt cutter, splitting shears, and a 1½-ton air hammer, together with forges, and all ordinary tools of an up-to-date shop, including a portable oxyacetylene welding outfit. There is also a carpenter shop and an electrical shop, where motors are rewound and general repairs made.

The mechanical crew consisted of a master mechanic, one machinist, one blacksmith, one carpenter, two electricians, one blacksmith helper, one carpenter helper, and seven machinist helpers, making a total of 15 men to keep the entire plant in repair.

VIII. WATER SYSTEM

A concrete reservoir 144 ft. above the converter floor, holding 150,000 gal. in reserve in case of fire or break-outs around the furnaces, is kept filled by a two-stage turbine pump directly connected to a 100-hp., 550-volt, induction motor, located on Boundary Creek.

Water from Copper Creek is caught in a pond, from which it is pumped through a 14-in. steel main to a set of three tanks of 220,000 gal. total capacity, located above the furnaces, by two of three direct-connected Byron-Jackson centrifugal pumps operated by one 50-hp., 550-volt, induction motor, and two 40-hp., 550-volt, induction motors, respectively. The water from these tanks was used for jacket cooling, which required about 2,500 gal. per minute, and was then returned to

the pond, where, under normal conditions, it cooled off enough to be used over again, but in very warm weather or in winter when there was practically no cold water entering the pond from Copper Creek it was necessary to cool the water in the tanks by the addition of water pumped from Boundary Creek by a single-stage turbine pump directly connected to a 50-hp., 550-volt, induction motor. The piping of the water system is arranged as far as possible in independent circuits, to allow the shutting off of any section for repairs.

IX. ASSAY OFFICE

A fully equipped assay office, for handling the ores received at the smelter, as well as the slags, mattes, and blister copper, is situated in the general office building, a short distance to the east of the smelter proper.

X. APPENDIX

The following tables and the drawings will give a better idea of the foregoing description:

A. BRITISH COLUMBIA COPPER CO. SMELTER. AVERAGE DAILY REPORT
FOR MAY, 1913. TWO FURNACES, NOS. 1 AND 3*Labor*

FURNACES	Foremen	Furnaces	Charge Track	Slag Track	Brig. Mill	Power House	Totals
Shift 1.....	1	4	8	2	4	1	20
Shift 2.....	1	4	8	2	..	1	16
Shift 3.....	1	4	8	2	..	1	16

CONVERTERS	Foremen	Converters	Crane	Labor	Lining	
Shift 1.....	1	4	2	3	5	15

SAMPLING	Foremen	Sample Mill	Bins	Bucker	
Shift 1.....	1	2	5	1	9

SURFACE	Warehouse	Slag Track	Office Boy	Chauffeur	Janitor	Nipper	
Shift 1.....	2	3	1	1	1	1	9

MECHANICAL DEPARTMENT	Foremen	Blacksmith Shop	Machinist	Machinist Helpers	Carpenters	Electri- cians	
Day Shift.....	1	2	1	9	1	2	16

B. SMELTER OPERATIONS. AVERAGE DAILY REPORT FOR A MONTH WHEN THREE FURNACES WERE RUNNING

Delays: No. 3 was down 4 hr. on the 12th for repairs, and 8 hr. on the 21st, short of ore.

Labor

FURNACES	Foremen	Furnaces	Charge Track	Slag Track	Briquet Mill	Power House	Totals
Shift 1.....	1	6	11	3	4	1	26
Shift 2.....	1	6	11	3	..	1	22
Shift 3.....	1	6	11	3	..	1	22

CONVERTERS	Foremen	Converters	Crane	Labor	Lining	
Shift 1.....	1	2	2	3	5	13
Shift 2.....	..	2	2	1	3	8

SURFACE	Warehouse	Slag Track	Office Boy	Chauffeur	Janitor	Nipper	
Shift 1.....	2	4	1	1	1	1	10

SAMPLING	Foremen	Sample Mill	Bins	Buckers	
Shift 1.....	1	3	6	2	12

MECHANICAL	Foremen	Blacksmith Shop	Machinist	Helpers	Carpenters	Electri- cians	
	1	2	1	9	2	2	17

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Metallurgical Practice in the Witwatersrand District

Additional discussion of the paper of F. L. Bosqui, presented at the San Francisco meeting, September, 1915, and printed on pp. 24 to 81 of this volume.

A. L. BLOMFIELD, Denver, Col. (communication to the Secretary.*)—Mr. Caldecott says on p. 67: "The Dorr thickener shown, while a useful device when crushing with cyanide solution for removing surplus solution from slime pulp prior to air-lift agitation or vacuum filtration, is not adapted to effect as complete a separation of water from settled slime as the ordinary intermittent settlement and decantation."

This statement might have passed muster in this country several years ago when the Dorr thickener was a new device, but its use in hundreds of mills in the United States has demonstrated that as thick a product is being discharged from it as was obtained from the intermittent settling which it has supplanted.

It is interesting to note that recent investigation, especially the work of H. S. Coe and G. H. Clevenger,¹ in the March *Bulletin* of the Institute, under the title: Laboratory Method for Determining the Capacities of Slime Settling Tanks, has developed the general theory of settlement and explains the results obtained in practice.

Just as free settlement is a function of surface area, so thickening after the zone of free settlement is passed is a function of time. The time required is obtained by the capacity or volume provided in the settler for this thickening zone. Each pulp is a problem in itself, both in regard to free settlement and to time required to reach a commercially finished thickness. Some pulps require 48 hr. and more to reach a density of 40 per cent. solids; others no coarser to the screen test reach a density of 65 per cent. solids in a fraction of that time. In thickening any pulp a point is reached where the increase in density is so slow as to be commercially unprofitable. When any part of the pulp has reached this point it should pass on for treatment—its place in the thickener should be occupied by pulp which has not yet reached the commercial limit. By doing this the whole volume of the settling tank is kept in active use. Thus the continuous thickener keeps the whole tank at work to its commercial limit, while the intermittent settler has much of its volume occupied at times by pulp which is either quite finished thickening or at least thickened past the commercial point.

The mechanical efficiency of the continuous thickener can be the only argument for limiting this statement. In my own experience I have never

* Received Mar. 13, 1916.

found the thickness to which a pulp could be reduced limited by the inability of the Dorr settler to collect and discharge the pulp as thickened; the limit has always been imposed by the pulp itself reaching a point where increase in density was so slow as not to justify provision for greater thickening capacity. Certainly it is not exceptional in practice to find the continuous thickener regularly discharging pulp thicker than 60 per cent. solids.

Possibly the slight stirring of the rakes even aids thickening by breaking up the flocs in the thickening pulp at a point where it is difficult for reflocculation to take place.

As stated above, numbers of intermittent settling plants have been replaced by continuous settlers, resulting in increased capacity for given surface areas and tank volumes.

For instance, some six years ago I was operating six 30- by 10-ft. settlers intermittently, the object being to get as thick a pulp as possible, thereby obtaining the greatest possible change in solution before agitation in a 2:1 solution. As the tonnage required of the plant rose, the settling and thickening capacity became strained and we installed four Dorr devices in four of the 30- by 10-ft. tanks. The four continuous thickeners handled the settling and thickening as efficiently as the six intermittent settlers had, and at the same time: (1) Saved labor of decanting and sluicing; (2) saved labor of changing flows from one settler to another; (3) cut out the necessity for large intermittent additions to the mill circulating or crushing solutions which had been necessary while refilling each of the intermittent settlers as each in turn was sluiced out and turned back into the system; (4) kept a circulating solution in which chemical contents were most easily maintained constant; and (5) forwarded a continuous steady feed of thick pulp to the agitation plant.

Incidentally I might mention that these four thickeners have been converted into "connected type tray thickeners," converting the plant into a partial countercurrent decantation plant, before filtering, with excellent results. Each tray inserted has given a full 100 per cent. increase in capacity.

When Mr. Bosqui referred to the thickeners discharging a 50 per cent. pulp, quite possibly on the particular ore in question that was the commercial limit. On Mr. Caldecott's pulp the commercial limit is evidently 40 per cent. moisture and this is most certainly not the mechanical limit of the continuous thickener.

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